Chapter 1

Quarrying

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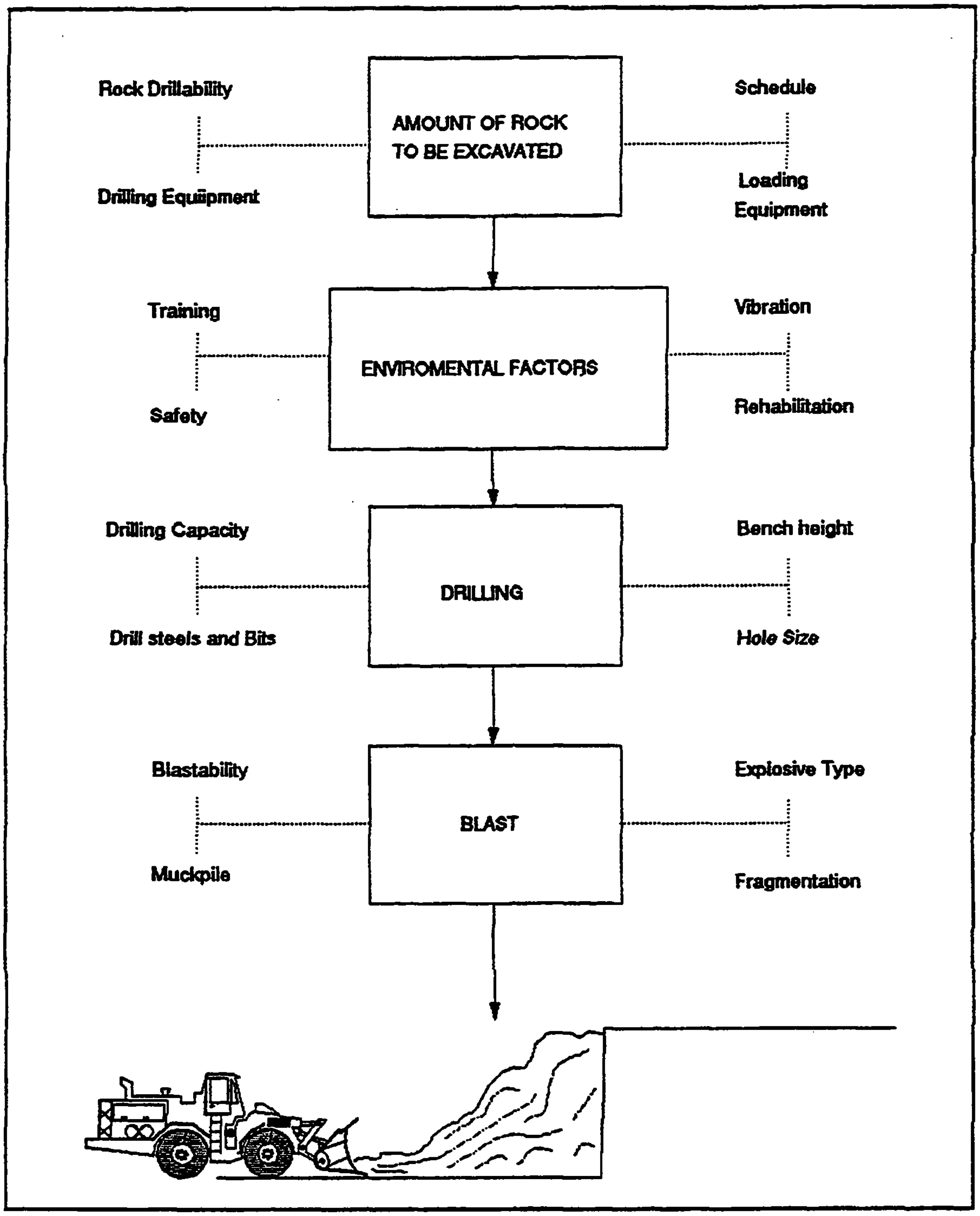
1. INTRODUCTION

In the cement industry quarrying is the mining method for the production of raw materials in the process of making cement. Quarrying describes the surface mining of rock whereas 'open pit' mining describes the surface mining of minerals. Discontinuous mining systems such as scraping, ripping and dozing and blasting are commonly practiced in limestone quarries. Continuous mechanized mining systems, with bucket wheel or chain excavators, are used where the deposits are consistent and soft such as chalk.

Quarrying is the breaking of the rock in a safe and economic way and transporting the result to a plant for further reduction in size. Figure 1 graphically sketches the sequence of events required to bring quarry into operation.

This involves planning, purchasing of suitable equipment, drilling, blasting, loading and transport of rock in quantities sufficient to permit the continuous operation of the cement plant. All this must be accomplished as efficiently and safely as possible to maximize the return on investment. Finally environmental considerations must be remembered and careful control kept on noise and dust levels. Parts of the quarry that become worked out must be rehabilitated so that there are few scars on the landscape.

Figure 1: Quarry Operation



Blasting is the most widely used method to excavate limestone for cement production as the rock is usually too hard to be ripped or dozed. This involves the drilling of holes in the rock, placing a predetermined charge of explosive in each hole and detonating it. The result is a pile of broken rock which then has to be removed to the cement plant. As it is not economic to break each piece of rock to the required dimensions in the blasting process the rock is further reduce in size before being transported to the cement plant storage silo. This further reduction is accomplished by means of a crusher system.

To obtain optimum results from the quarry operation the rock to be blasted has to be matched to the explosive and the drilling parameters. Resulting fragmentation and muckpile shape, broken rock from the blast, is dependent on all three factors.

2. GEOLOGY AND GEOGRAPHY

There are many rock types existing in the earth's crust and all are due to geological processes that started about 4,5 billion years ago. By contrast man has only been in existence for a few thousand years.

2.1 Rock Hardness

Rock hardness is classified by using 'Mohs scale of hardness'. This lists ten rocks with differing degrees of hardness in an ascending order. The rocks used as standards, their Mohs' hardness and basic classification are to be found in table 1.

Limestone, with a high percentage of calcium carbonate, is one of the sedimentary rocks found in the crust of the earth. It has an average Mohs's hardness of about 3.3. This means that it is usually necessary to blast limestone to free it from the surrounding rock.

Table 1 Moh's scale of hardness

Rock	Hardness	UCS (MPa)	
Talc	1	-10	soft
Gypsum	2	10	
Calcite	3	30	
Fluorite	4		medium
Apatite	5		
Feldspar	6	120	
Quartz	7		hard
Topaz	8	200+	
Corundum	9		very hard
Diamond	10		

2.2 Influence of Rock Characteristics

There are a number of theories relating to the effect that geology and in particular rock hardness has on the breaking effect of explosives. In most cases results are based on the assumption that the ground is homogeneous and contains no jointing bedding or planes. However these theories can be used as a guide as to the hole burden and spacing necessary to give the required fragmentation when using a particular explosive. They remove some of the guesswork and trial and error from the planning. To determine these factors it is necessary to obtain certain rock properties. This can be partly accomplished in the laboratory and partly by examination of core samples. The properties required are:

Density	- specific weight
Young's Modulus	- stress to strain ratio
Poisson's Ratio	- measure of elasticity
Uniaxial Compressive Strength	- static load necessary to break a sample of rock

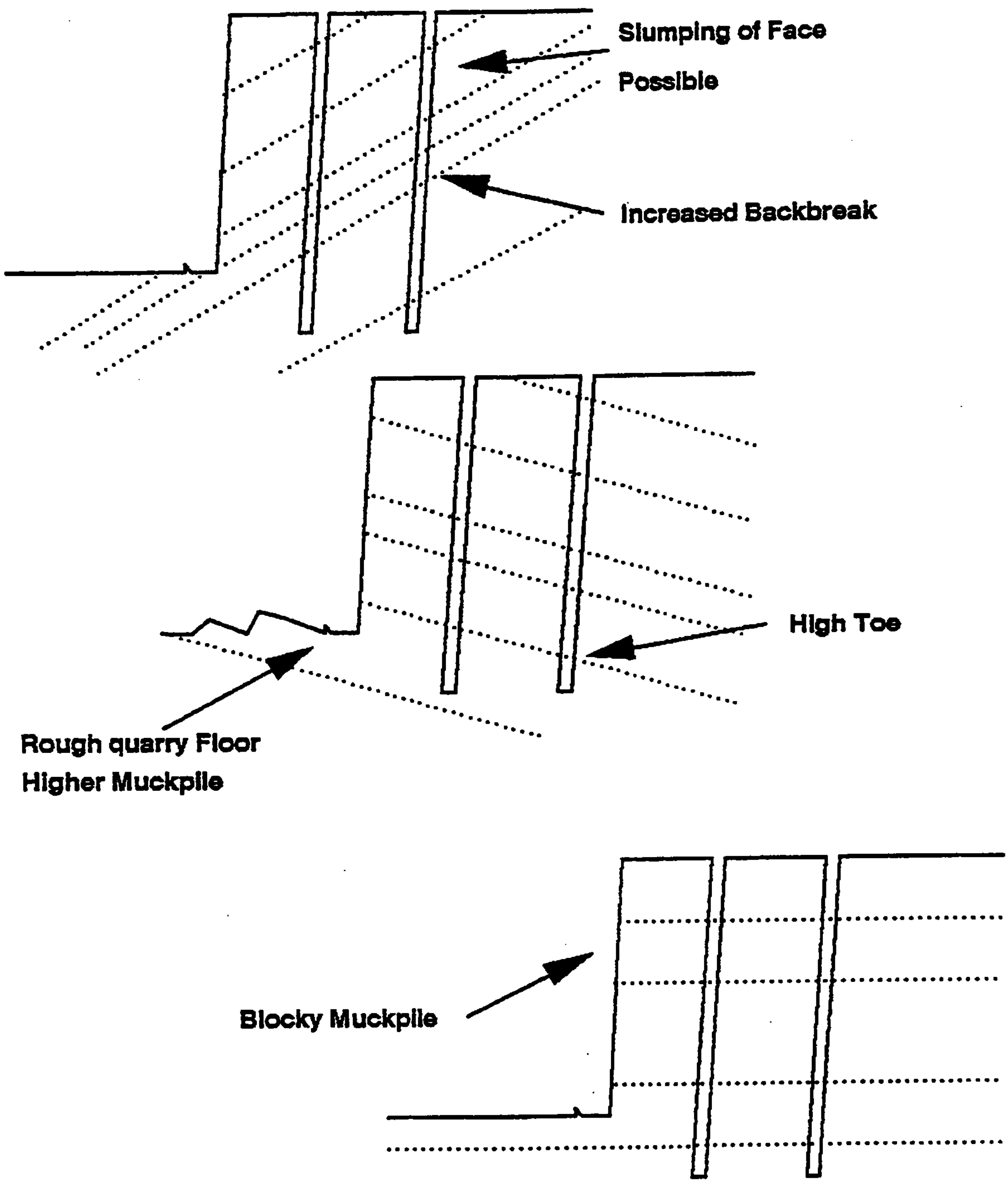
Results from these tests can give a guide as to the best explosive and the correct drill hole diameter to provide the most efficient drilling and blasting at the cheapest cost.

2.3 Influence on the Blasting Process

Geology, in the system of jointing and bedding planes, will dictate the bench geometry and face orientation as well as the resulting fragmentation. Examples of the influence of dip can be seen in figure 2 which shows quarry faces with rock formations both dipping into and out of the face.

Jointing has the same effect on drilling and blasting as schistosity and bedding planes. The most significant property of a joint is its inability to transmit tensile stress. Its tensile strength can be considered as zero, or minute in comparison with solid rock. Stress waves are reflected from joint walls and therefore interrupted in their travel through the rock. The visible result of this is large, blocky boulders in the muckpile.

Figure 2 Jointing Effect on Breaking



2.4 Topography

Geography will be considered when planning a new quarry. The topography of the country will determine the development of the quarry, pit like operations will result in flat terrain. Contour or side hill operations would be found in hilly areas. Accessibility to the area, roads, services, and buildings will also determine the development and infrastructure required at the quarry site. Additional problems are created when the proposed quarry is close to a built up area. Complaints can be expected by local inhabitants, from blasting and the noise made by machinery. The proposed area of a quarry may be limited by its proximity to an area of natural beauty. Environmental damage to the area and the building of access roads have to be minimized if the quarry is in a tourist area.

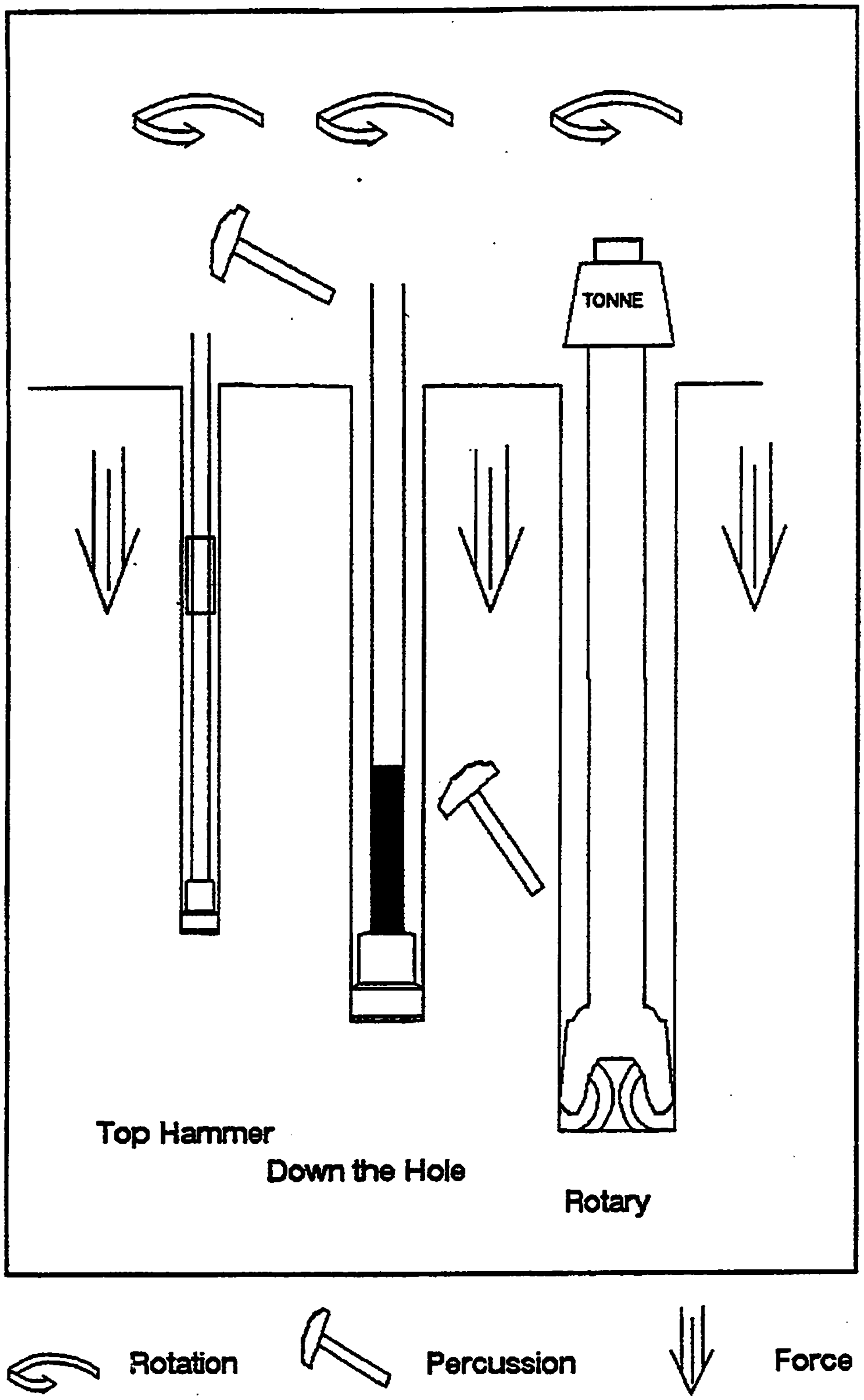
3. DRILLING

Bench drilling is a term for designating the method where surface holes are drilled for blasting towards a free face. Blasted rock then has ample space in which to expand. Bench heights vary between 5,0 meters and 30,0 meters, depending on the thickness of the formation, however the limiting factor in the height of the bench is usually that of safety or hole diameter. Safety can be the inherent stability of the formation, the greater the stability the higher the bench and hence the longer the drill hole. An increase in the stability of the formation can be achieved by the use of inclined drill holes. This creates an artificial slope at an angle of between seventy and eighty degrees. If a failure of the bench occurs the rock will tend to roll down the incline face and not drop thus confining loose rock.

Larger hole diameters mean that longer holes can be accurately drilled. Toe position of a hole is important if successful breaking is to occur. With hole diameters of 64 mm or less a hole drilled over 20 meters is likely to wander from the line. Requirements of a drillhole are drawn in figure 3.

Quarry operations usually use an intermediate diameter drillhole of between 64 mm and 165 mm although very large quarries can have hole diameters of up to 440 mm.

Figure 3 Drilling Methods



3.1 Drilling Methods

3.1.1 Rotary Drilling

Most commonly used in soft-medium rock masses that have an average Mohs' hardness of less than about four. This is a drilling method that was originally used for oil well holes. In rotary drilling energy is transmitted via a drill rod which is rotated at the same time as the drill bit is forced down. The bit is rotated continuously at between 50 and 90 revolutions per minute. Downward thrust is achieved by the weight of the machine. This force is used to push the roller inserts, the cutting edges on the bit, into the rock which on rotation break off small chips of the rock. The relationship between feed force and rotation rate determines drilling efficiency. Energy losses in rods are minimal in rotary drilling. All such drilling requires high feed pressure and slow rotation but the relationship varies with rock type. Softer rock requires lower pressure and higher rotation speed and vice versa.

3.1.2 Percussion Drilling

Practiced where the rock is hard to very hard and abrasive, divided into:

3.1.2.1 Top Hammer Drilling

A shank adapter is hit repeatedly by a piston in the drilling machine which creates a shock wave that travels down the drill string to the bit. Energy is discharged against the bottom of the hole and the surface of the rock is crushed. Air traveling down the center of the drill string flushes the resulting chips out of the hole. The whole drill string is rotated and crushes each segment of the hole face in turn. Drilling machine and string are arranged on a feed, usually either chain or screw, and the feed force can be kept in contact with the hole bottom by adding extra drill rods when the feed mechanism reaches the limit of its travel.

3.1.2.2 Down the Hole Hammer

The hammer and its impact mechanism operate down the hole. The piston striking directly on to the bit. No energy is therefore lost through transmitting the shock wave down a drill string. The drill tubes carry the compressed air for hammer and flushing.

3.1.2.3 Overburden Drilling

A new quarry operation is fortunate if the rock to be blasted is fully exposed on surface. In the majority of cases it is overlain with soil and weathered, powdery rock called collectively overburden. This must first be removed before the rock below can be removed. In cases where overburden is soft dozers and rippers can accomplish the work.

Conventional methods, as described above, are not suited to these ground conditions and problems could be experienced with stuck rods. A special drill that has casing tubes with its own ring bit, in addition to the normal bit, are manufactured for these conditions. This specially designed drilling rig has the following attributes:

1) Drilling will be continuous through varying ground conditions.
2) If stopped drilling can continue in loose ground easily.
3) The hole is prevented from collapsing when the bit is removed.
4) Flushing is efficient and rotation torque is high.

3.2 Comparison of Drill System

Penetration rate of a down the hole drill is, in theory, independent of hole length and therefore will be capable of drilling longer holes with less deflection than the surface hammer. As the rotation mechanism is down the hole these drills perform better in fractured ground than the surface drills. The rods of the latter may jam when rock fragments fall down the hole.

Down the hole hammers, because of their mechanism, are limited in the size of hole they can drill, the minimum being about 92 mm. For smaller diameter a surface percussion drill will be used. Rotary drilling is the major method of blast hole drilling in large limestone quarries, where hole diameters of up to 150 mm are found. Hydraulic drilling has advantages over pneumatic in that the transmission of forces and energy is accomplished by the use of a fluid. Fluids are virtually incompressible and only a fraction of the energy is lost in transmission. Together with a high rotational speed this permits a higher drilling rate.

The machines used for rock drilling can be roughly grouped into the following based on their working principles:

3.2.1 Top Hammer

- Hydraulic rock drills
- Pneumatic rock drills

3.2.2 Down the Hole

- Pneumatic hammer and pneumatic rotation unit
- Pneumatic hammer and hydraulic rotation unit

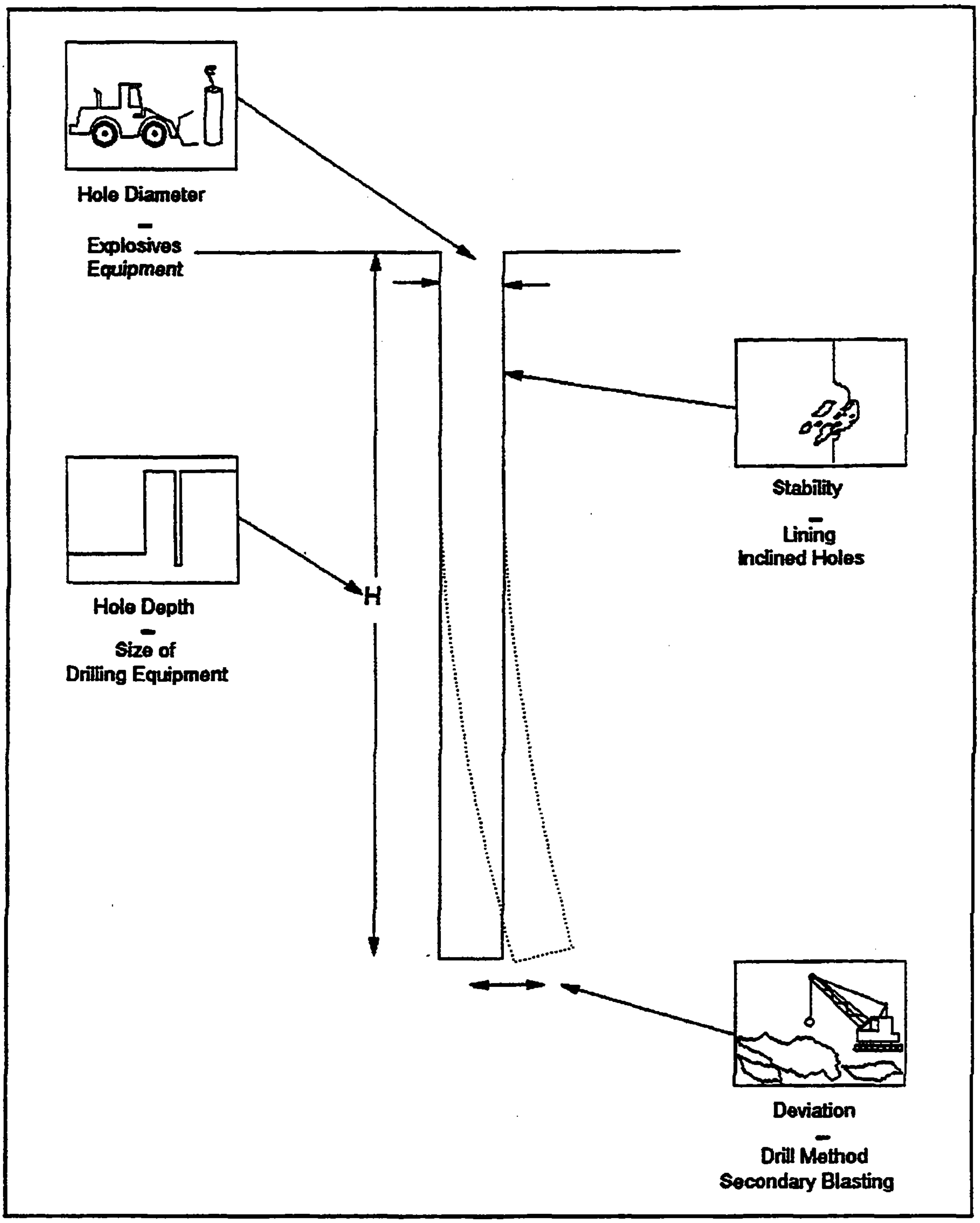
3.2.3 Rotary

- Small and large hole rotary cutting
- Large hole rotary crushing

3.3 Drilling Parameters

Determining the size and number of blastholes as well as the drill rig required to drill them is based on many different parameters. Some of these demands are shown in figure 4.

Figure 4 Demands on Drilling



Other demands include:

3.3.1 Capital

Cost outlay required to purchase rigs.

3.3.2 Nature of the Rock

Jointing, bedding planes, a smaller hole diameter will permit better distribution of the explosive in the rock and a better explosive efficiency. A larger hole diameter means a larger burden and more concentration of the explosive. Although drilling costs would be lower, fewer joint bound blocks would be intersected by drill holes resulting in excessive secondary blasting costs.

3.3.3 Fragmentation Size Distribution

Efficient drilling designs, combined with the correct choice of explosive result in better fragmentation. Correct collaring of holes improves the whole quarry efficiency. If holes are drilled out of position or of the incorrect length then fragmentation is going to be poorer. Too long holes will result in wander of the toe of the hole with resulting increases in burden and spacing again resulting in poor fragmentation and overall explosive performance.

3.3.4 Monthly tonnage requirements

Larger hole diameters are used where there is a greater tonnage requirement.

3.3.5 Cap Rock

Problems with oversize from the collar area. Smaller diameter holes require a shorter stemming length and higher explosive column in the hole.

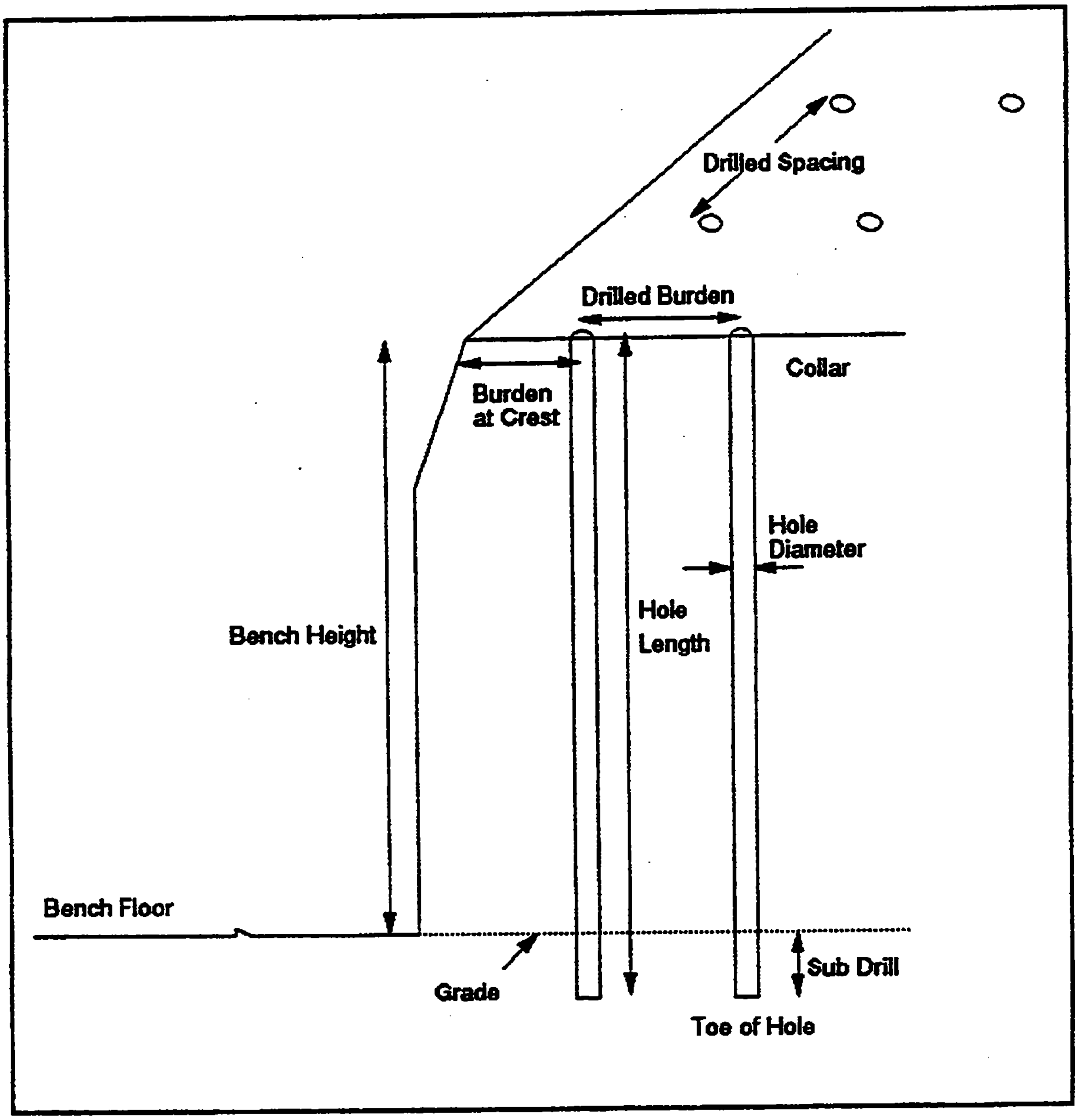
3.3.6 Vibration and Airblast

Smaller hole diameters imply a smaller mass of explosive per delay reducing the level of ground vibrations.

3.4 Drilling Definitions and Equations

The relationship of the various parameters used in drilling is shown in figure 5. There are no 'hard and fast' rules. They only serve as a guideline and are based on experience.

Figure 5 Drilling Technology



3.4.1 Borehole Diameter

This is important to obtain maximum fragmentation at the lowest cost. The cost of drilling, per cubic meter blasted, decreases as the hole diameter increases. However too large a diameter can cause problems with resulting airblast and flyrock if the bench height is too low. Excessive jointing at small intervals in the rock can cause a fragmentation problem if a large hole diameter is selected. For best fragmentation control the appropriate bench height to borehole diameter is 0.12 to 1 (in meters and mm). That is:

0.12 * D	=	H
where D	=	borehole diameter in mm
H	=	bench height in meters

3.4.2 Burden

This is considered the most critical variable in the design of surface blasts. It is defined as the distance from a borehole to the nearest free face at the time of detonation. In planning, the burden is taken as the distance at right angles to the free face, from the free face to the first row of blastholes. Burden is a function of the charge diameter and therefore also depends on the drillhole diameter.

B	=	(25 to 35) * E
where B	=	burden in meters
E	=	explosive in hole, diameter in meters

3.4.3 Spacing

Is the distance between adjacent boreholes and is measured parallel to the free face. The optimum spacing to burden ratio is between 1 and 1.3. Too small a ratio leads to holes not breaking out, with resulting flyrock and ground vibration problems. Too large a ratio has the effect of increasing oversize as the blastholes break individually to the free face.

3.4.4 Bench Height

Distance, measured vertically from one level floor of a quarry to the next up or down. For a successful design it is important that the burden and bench height are reasonably compatible. The minimum is:

H	=	2 * B
where H	=	bench height in meters
B	=	Burden in meters

If the bench height is low and the burden and blasthole is large then an excessive percentage of the borehole is taken up with stemming. Very high benches become a danger for personnel working beneath them.

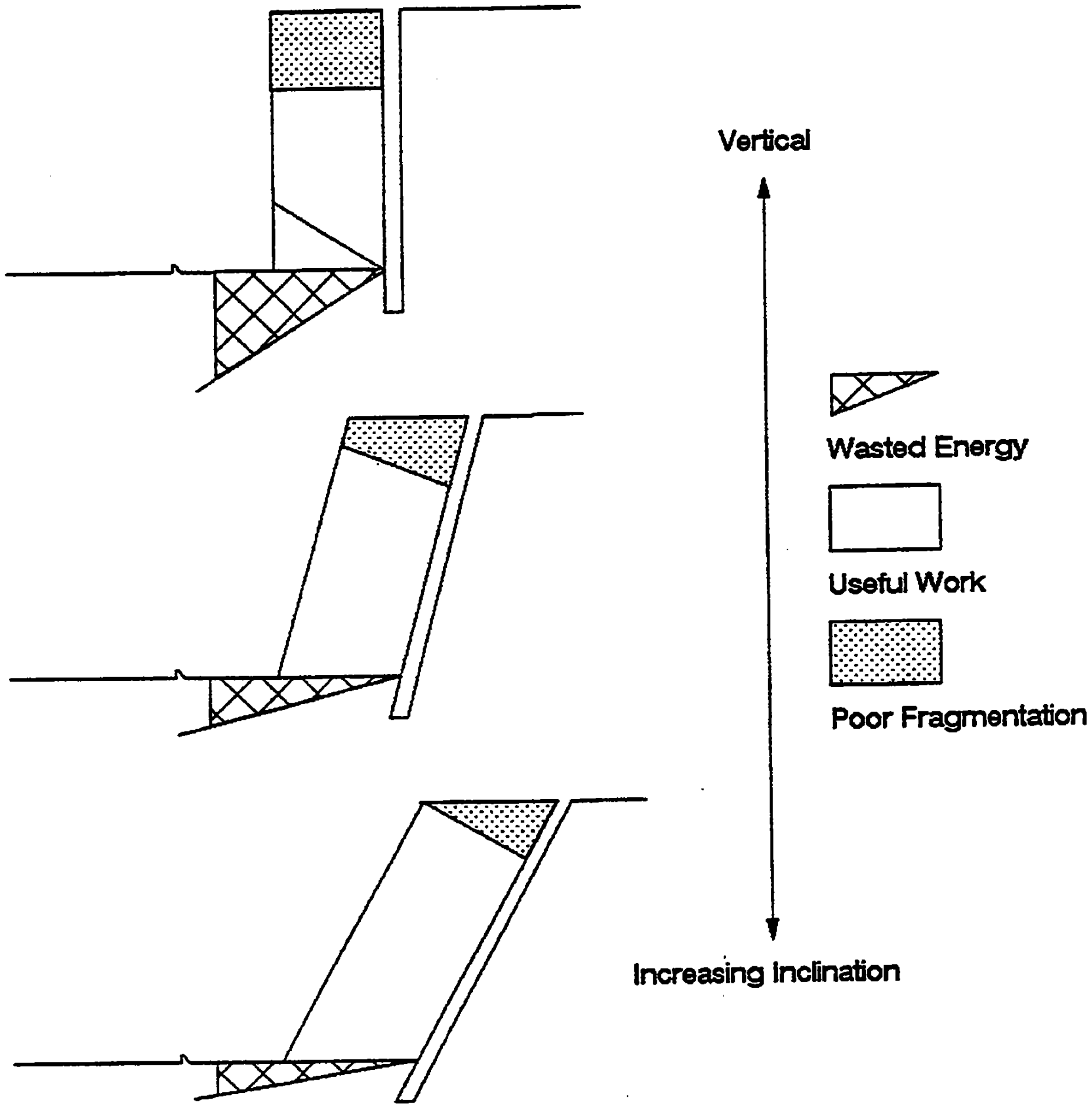
3.4.5 Subdrill

The distance drilled below the level of the bench floor necessary to be certain that, on blasting, the toe of the holes breaks out.

3.4.6 Vertical and Inclined Holes

Drill holes are drilled either at an angle to the vertical in the direction of the free face or vertical. There are several advantages to incline drilling as detailed in figure 6. In addition there can be a reduced explosive cost due to increased burden. There is also less risk of backbreak and toe problems. However inclined drilling needs closer supervision in order to achieve good results. If the hole is underburdened then flyrock is more likely from blowouts.

Figure 6 Effect on Inclining Holes



3.4.7 Drill Hole Deviation or Wander

The drill bit and drill string can be deflected from its planned straight course after collaring. The major causes are:

3.4.8 Collaring

The start of the hole where the drill bit is placed on a marked position and drilling commences.

3.5 Drilling Patterns

Blast holes are drilled to a pre-planned pattern which will determine:

- number of holes and drilled meters
- cubic meters of rock to be blasted

Drilling plans are either made using prepared drilling tables for certain parameters or they can be made especially with a desired result in mind.

The most common pattern are:

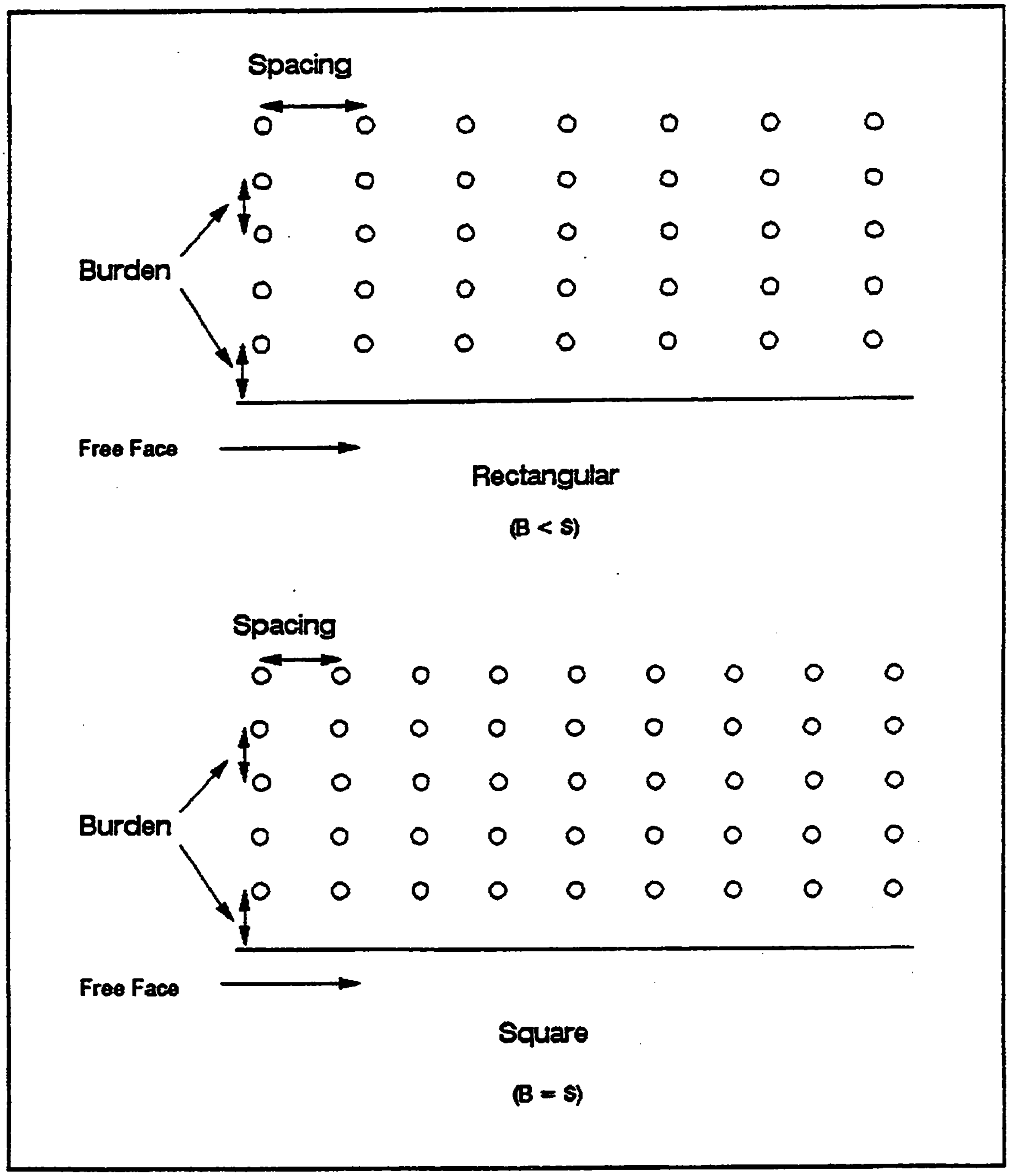
3.5.1 Square

Equal burden and spacing

3.5.2 Rectangular

Burden is less than the spacing.
Easily marked out and easy to collar accurately.
Figure 7 shows these patterns.

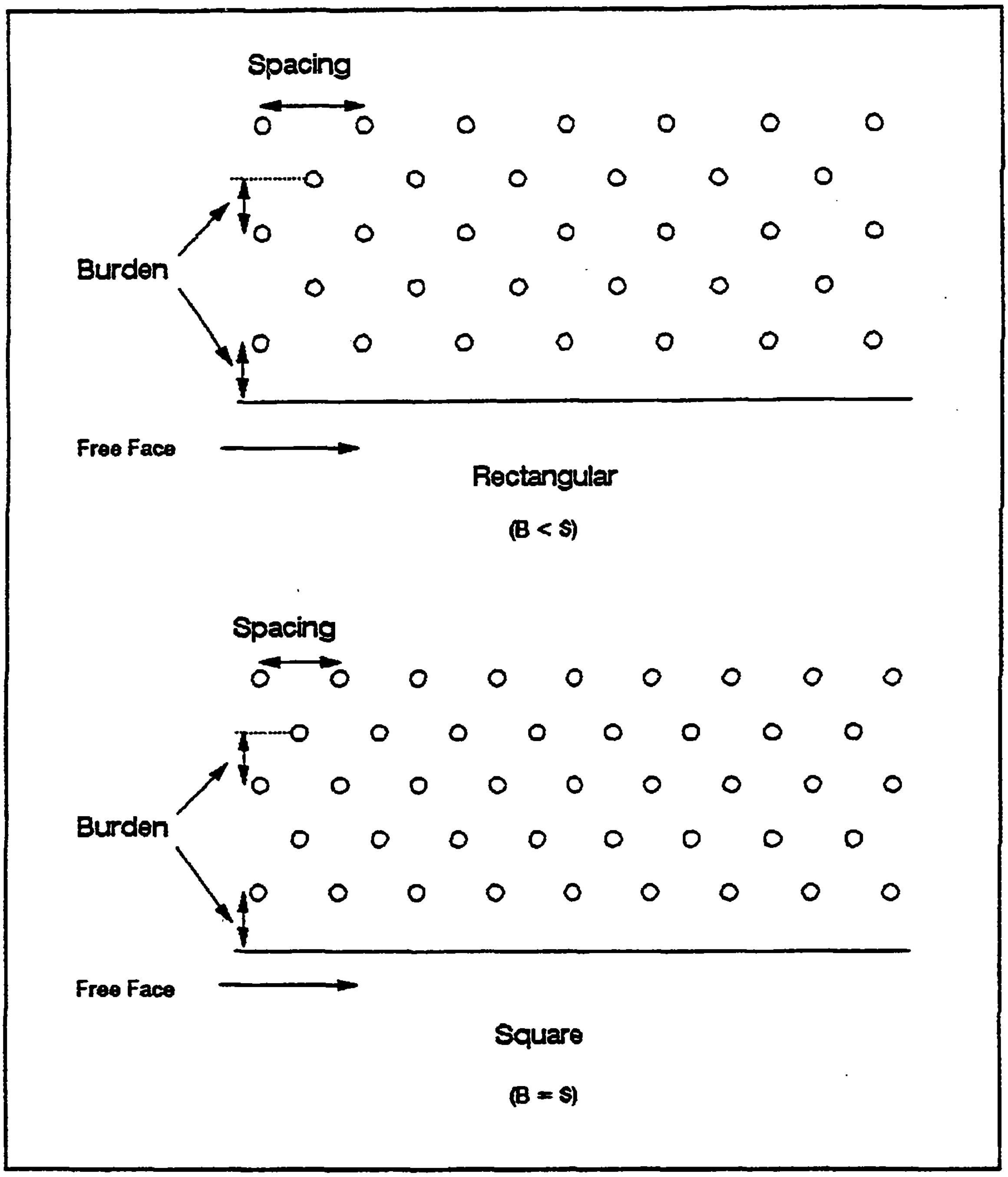
Figure 7 Inline Drilling Patterns



3.5.3 Staggered

Spacing to burden ratio may be one is more usually greater than one. The holes in alternative rows are in the middle of the spacings of the row in front. Figure 8 shows an example of each pattern.

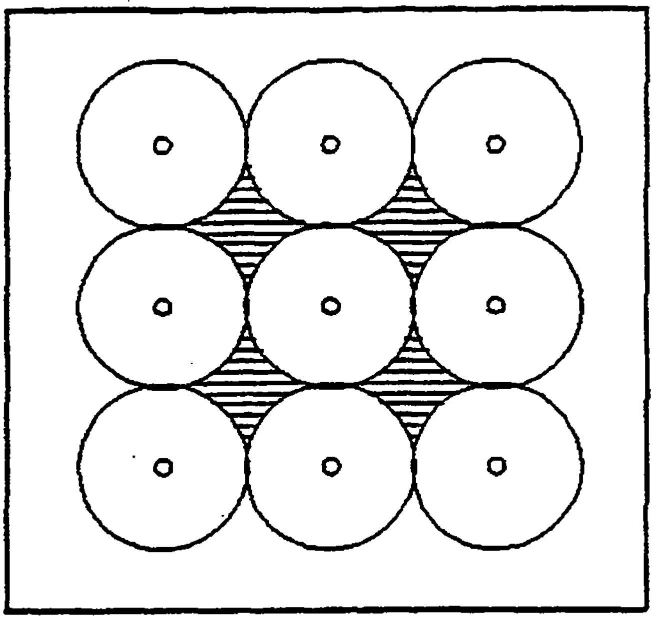
Figure 8 Drilling Patterns for Staggered Holes



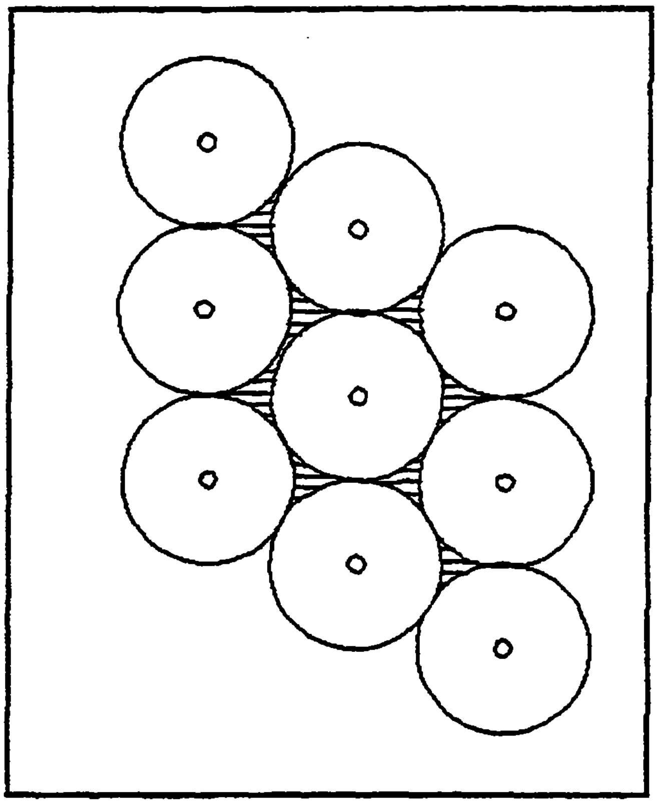
Ideally holes should be drilled in a triangular pattern where the spacing to burden ratio is 1,15. This equilateral triangular pattern provides for the optimum distribution of the explosive charge, for any particular hole diameter, throughout the rock. In this way the cost of drilling and blasting can be optimized. The area around the hole influenced by the detonation of the explosive and the distribution of the resulting explosive energy throughout the surrounding rock is graphically shown in figure 9 and compared to other patterns.

Figure 9 Comparison of areas of Explosive Influence around a Blasthole

Inline Pattern



Staggered Pattern



4. BLASTING

Blasting is an intermediate step in the process of quarrying. Choosing the correct explosives is one of the most important decisions influencing the design and operation of a quarry. It cannot be taken in isolation as other factors, such as rock type and hole diameter, also influence the choice. The major objectives of a blast are to suitably fragment the rock and to displace it such that it is easy and safe to load out. Once a blast has been initiated it is uncontrollable and cannot be repeated if incorrect. The greater the effort put into the planning and preparation the better the results will be.

4.1 History of Explosives

Commercial explosives started with the invention of black powder. The first recorded mention of 'saltpeter' is in the 13th century in Arabia. Its first recorded use in mining was in the Royal Mines in Hungary and from here its use spread to the tin mines of Cornwall in England in the late 17th century.

Ascanio Sobero discovered Nitroglycerin (NG) in 1846 and it was used in its raw state in blast holes with black powder igniters. This proved to be very hazardous and Alfred Nobel, while seeking a safer way to transport this new explosive chanced on Kieselguhr which rendered nitroglycerin less sensitive to shock, this was the first dynamite. Dynamite was not only much more powerful than black powder but with its higher velocity of detonation (VOD) it was more effective in breaking rock. The only real drawback was that it was not waterproof. In 1875 Nobel found that by dissolving nitrocellulose in nitroglycerin he could increase the water resistance of his explosive, which was called 'blasting gelatin'. The next step in the making of NG explosives was to replace part of the expensive nitroglycerin with low cost Ammonium Nitrate. It has only about 70 % of the blasting strength of NG and was highly hygroscopic but research to increase the percentage used in Dynamites continued. This resulted in a relatively cheap high performance explosive with good waterproof characteristics still in use today.

Concurrently better and safer ways to initiate the explosive cartridges were being discovered. In 1831 a Cornishamn invented Safety Fuse, a continuous core of black powder wrapped in jute and twine and coated with varnish to make it waterproof. However, this fuse would not initiate Nobel's explosives and he solved the problem with the patenting of his Fulminate of Mercurycaps. This also showed that to maximize the energy produced from a detonating explosive a shock wave was required. With this combination a certain reliability in timing and detonation was introduced at the initiation of explosives. Bridge wires and electric blasting were patented in the late 1800's and together with detonator delays produced superior fragmentation.

Devastating explosions in two ships in 1948, when fuel oil leaked into their cargoes of Ammonium Nitrate (AN), led to the discovery of the cheapest of all explosives. If AN is mixed with about 6 % Fuel Oil (FO) it becomes a powerful relatively safe explosive known as ANFO. AN is now manufactured in small prills to give the explosive good flow characteristics.

The increase in blasthole diameter in the 1950s led to new explosives coming into existence. These were the Watergels consisting of AN, water, a thickener and a sensitizer. Research had been conducted to waterproof AN products and it was found that the way to increase resistance to water was to shield it chemically. The main advantages were, a high loading density, good performance and low sensitivity. Above all they did not contain any headache causing ingredients, a big problem with all NG based explosives.

Emulsion explosives, developed in the 1960s, are a high performance explosive which detonates without a sensitizer being added. They are prepared in the form of oil in water emulsions. The basic ingredients are AN, water and fuel, the water in oil. A wide variety in water immiscible fluids permits a wide range of products, stiff to fluid. They are extremely stable over a wide temperature range.

Explosives today are defined either as 'poured', 'pumped' or 'cartriged'. Bulk loading of holes, from a mobile container often mixed to form an explosive on site, is safer and cheaper than loading cartridges into a hole. However where an exact charge per hole is necessary or small quantities required then packaged or poured explosive is preferred.

4.2 Breaking Process

An explosive is a chemical compound or mixture ignited by either shock, impact or friction. When ignited it decomposes rapidly as a detonation. There is a release of heat and large quantities of high pressure gases, which expand rapidly with sufficient force to overcome confining forces such as the rock around the drill hole. In commercial blasting the energy released by the detonation manifests itself in four ways, these are:

- rock fragmentation
- rock displacement
- ground vibration
- airblast

4.3 Explosive Properties

Each explosive has certain different characteristics or properties. Some of the major properties are listed below:

4.3.1 Velocity of Detonation

The speed at which a detonation wave travels through a column of explosives.

4.3.2 Density

Specific gravity, the standard being water.

4.3.3 Detonation Pressure

The pressure immediately behind the detonation front.

4.3.4 Energy

A measure of the potential of an explosive to do work.

4.3.5 Strength

The ability of an explosive to work.

4.3.6 Sensitivity

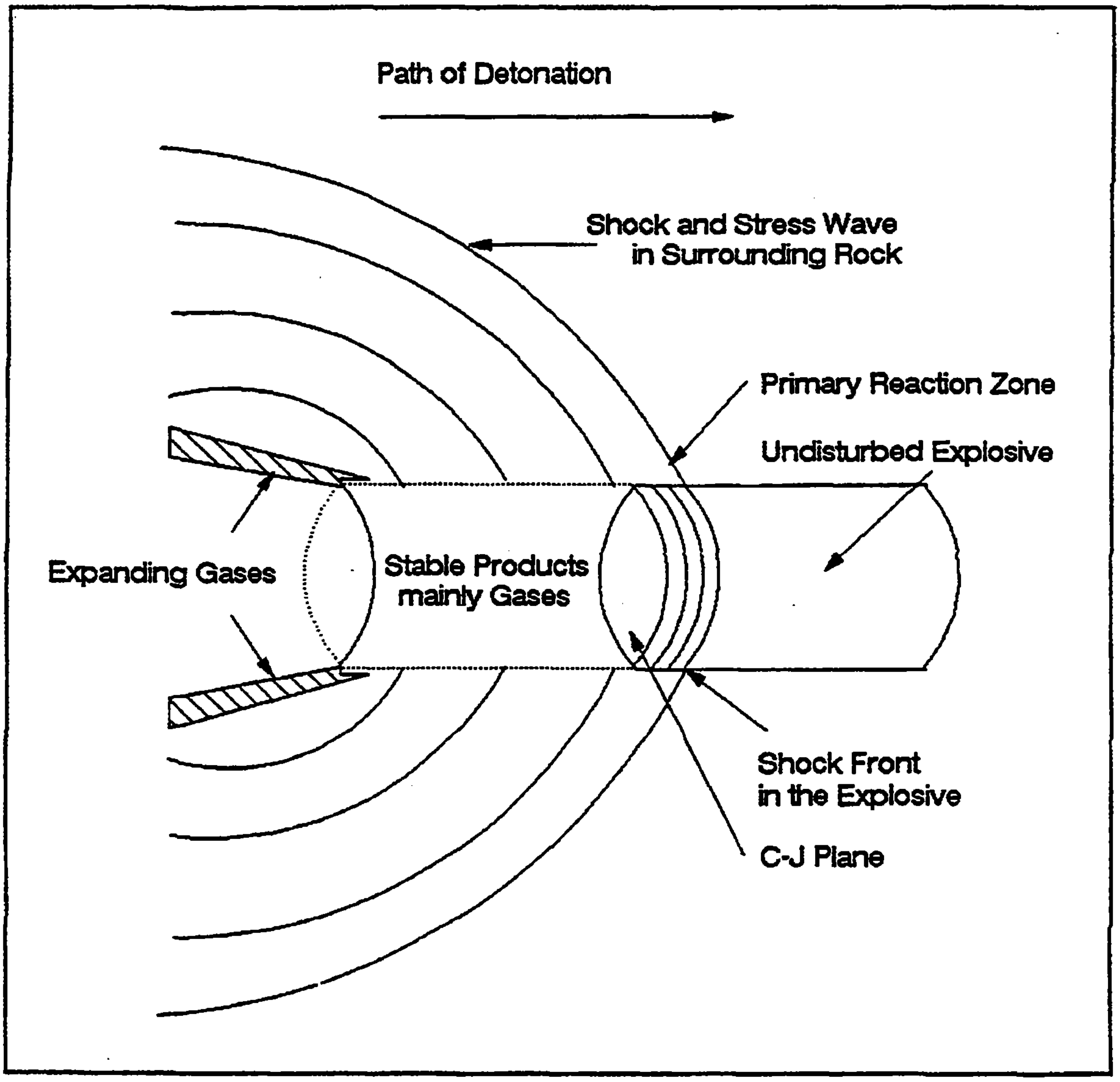
A measure of the minimum energy required to initiate the explosive.

4.4 The Process of Detonation

In a detonation the chemical reaction moves through the explosive material at a velocity greater than that of the speed of sound through the same material. The definitive characteristic if this chemical reaction is that it is initiated by, and supports, a supersonic shockwave proceeding through the explosive. Deflagration of an explosive occurs when the shockwave moves too slowly to produce significant shock energy.

Generally explosives with a lower VOD tend to release gas pressure over a longer period than those of higher VOD. These, lower VOD explosives, have more heave, important in areas where material movement is needed. Figure 10 shows a schematic drawing of a typical detonation.

Figure 10 Sketch of a Typical Detonation



In commercial explosives significant chemical reactions occur behind the primary reaction zone and effect the explosive performance. This is due in part to the need to make explosives safe to handle. The majority of products are gases at high temperature and pressure. In the order of 4000 degrees centigrade and 20 to 100 kilobars. These gases expand rapidly and produce the shock wave in the surrounding medium which in turn is transmitted into the rock around the borehole. Shock provides the energy for fragmenting the rock and gas, the heave energy to move the fragmented blocks to form the muckpile.

An explosion in a drilled hole is closely followed by the shock wave passing through the rock and stressing it, first in compression and then in tension. Tensile forces cause small radial cracks to develop from the hole which are subsequently expanded by the explosive gases entering them.

The free rock surface starts to move forward unloading the pressure. The tension increases in the primary cracks which expand to surface and complete the loosening of the rock.

4.5 Efficiency of Explosives

The efficiency of all explosives depends on several factors:

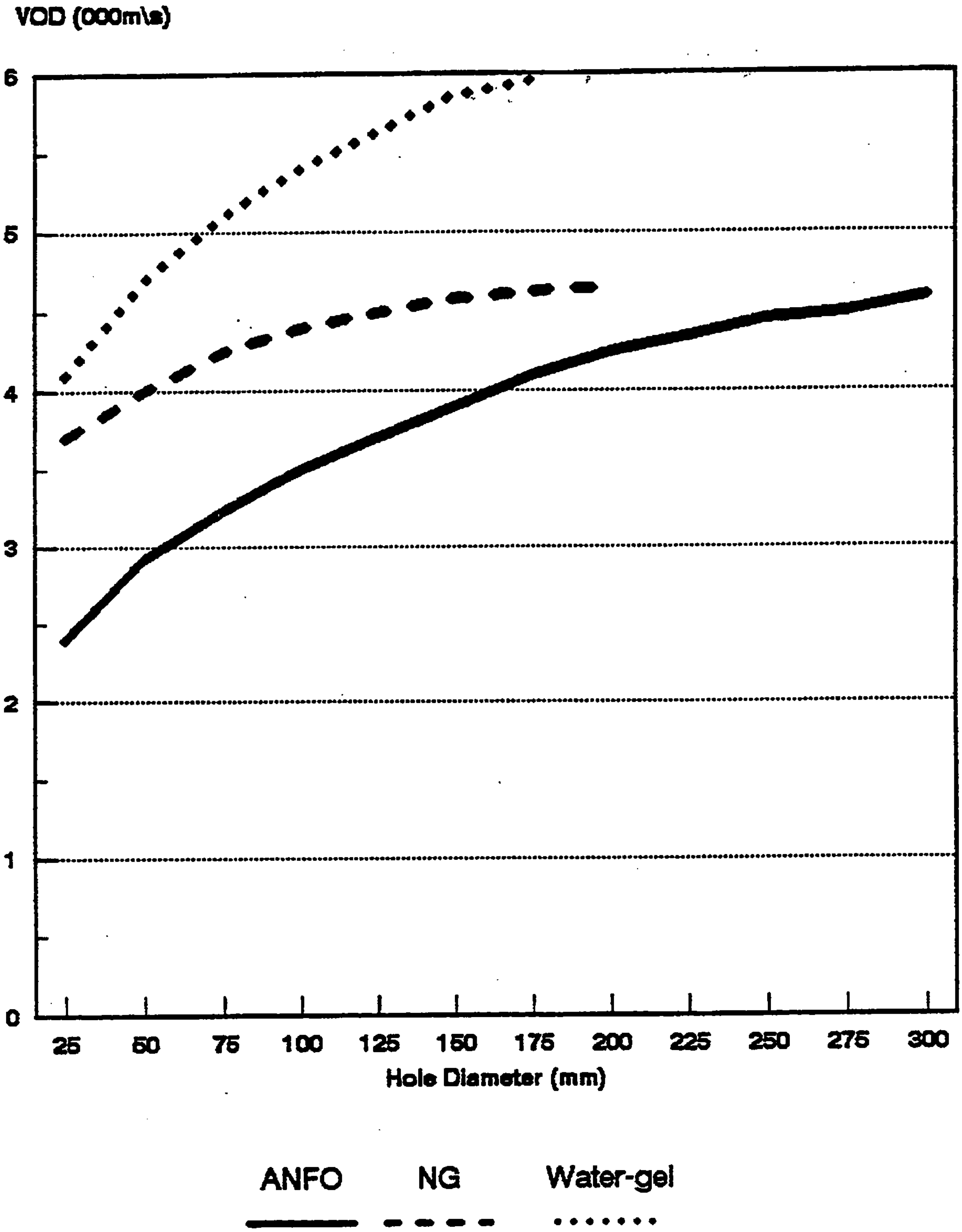
4.5.1 Coupling Ratio

The cross-sectional area of the drillhole filled with explosive. With pumped or pour loaded explosives this will be 100 %. Cartridges, not being able to fill the hole completely will always have a lower ratio.

4.5.2 Diameter

The larger the diameter of the borehole, which is the primary influence on the diameter of the explosive used, the greater the VOD of that charge. Figure 11 shows how the VOD of the common explosives is affected by the diameter.

Figure 11 Comparison of VOD with Hole Diameter



4.5.3 Priming

Adequate priming insures that the explosives will reach its maximum VOD as soon as possible. Inadequate priming can mean that the explosive efficiency is not fully exploited. A primer is an explosive, either a high VOD explosive or made from PETN, that accepts initiation from a detonator or detonating cord. It transmits this detonation to a mass of explosive contained in a drillhole. The cross section of the primer should be such as to match the hole diameter as closely as possible and as long as necessary to attain its maximum VOD. This will result in the detonation of the column of explosive at its highest potential to do work.

The primer should always be at the point of greatest confinement, if possible, at the toe of the hole. The exception is when horizontal band of harder rock is encountered in a series of drill holes. To break this effectively the primer can be located at this point in the hole.

4.5.4 Stemming

It is the non explosive material between the top of the explosive column and the collar of the hole. Stemming can consist of drill fines or gravel. Drill chippings are favored as they are readily available and their cost is minimal. The stemming height in a hole is dependent on the nature of the rock and amount of flyrock and noise that can be tolerated.

The use of effective stemming increases the amount of useful work that the explosive performs in the hole. This has a beneficial effect on the cost of the blasting operation.

4.6 Explosive Selection

Field and economic considerations influence the type of explosives needed to produce the desired results. Low cost, good fragmentation and adequate heave are the desired results. On the basis of cost no explosive can compete with bulk ANFO in dry holes. It accounts for about 80 % of all explosives used in surface blasting. Emulsion and ANFO emulsion mixes possess an increased efficiency value due to the fact that emulsion is a highly efficient explosive. Site conditions such as water, heave required and drilling cost can make the use of a water resistant, higher energy explosive, more attractive. The use of cartridged explosive in surface mining is unusual as the increased handling and storage costs required to place it into the hole makes it expensive. It has an application where a small, exact charge is needed in order to limit vibration or flyrock. Table 2 summarizes and compares the properties of the three most common explosive types.

Table 2 Comparison of properties

EXPLOSIVE	ANFO	WATERGEL	NG Based
Detonator Sensitive	No	No	Yes
Density	0,80	1,30	1,40
Relative Energy	1,00	1,82	1.66
Gas Development	34	39	38
Waterproof	No	Yes	Some-Yes
Bulk Loading	Yes	Yes	No

4.7 Powder Factor

A mathematical relationship between the weight of explosives and a quantity of rock expressed in kilograms, or grams, per ton broken. It is based on an assumption that explosive weight and explosive energy are one and the same. Different explosives have different energy outputs and cannot be compared to each other on this basis. To overcome this the term 'Energy Factor' has been introduced. In general the powder factor is related to the income producing unit of the operation.

4.8 Energy Factor

This is defined as the amount of explosive energy, in kilojoules, in a given quantity of rock.

Energy Factor = kJ/tons

5. INITIATION SYSTEMS

An initiation system is a combination of explosives and accessories designed to convey a signal to a column of explosives to detonate at a particular instant. The signal function may be either electric or non electric. Common systems by which explosives are detonated are:

- Initiation by an electrical impulse
- Detonating cord
- Safety fuse and plain detonator
- Initiation by a shock wave (Nonel)

The most widely used system is the electric detonator and associated circuitry.

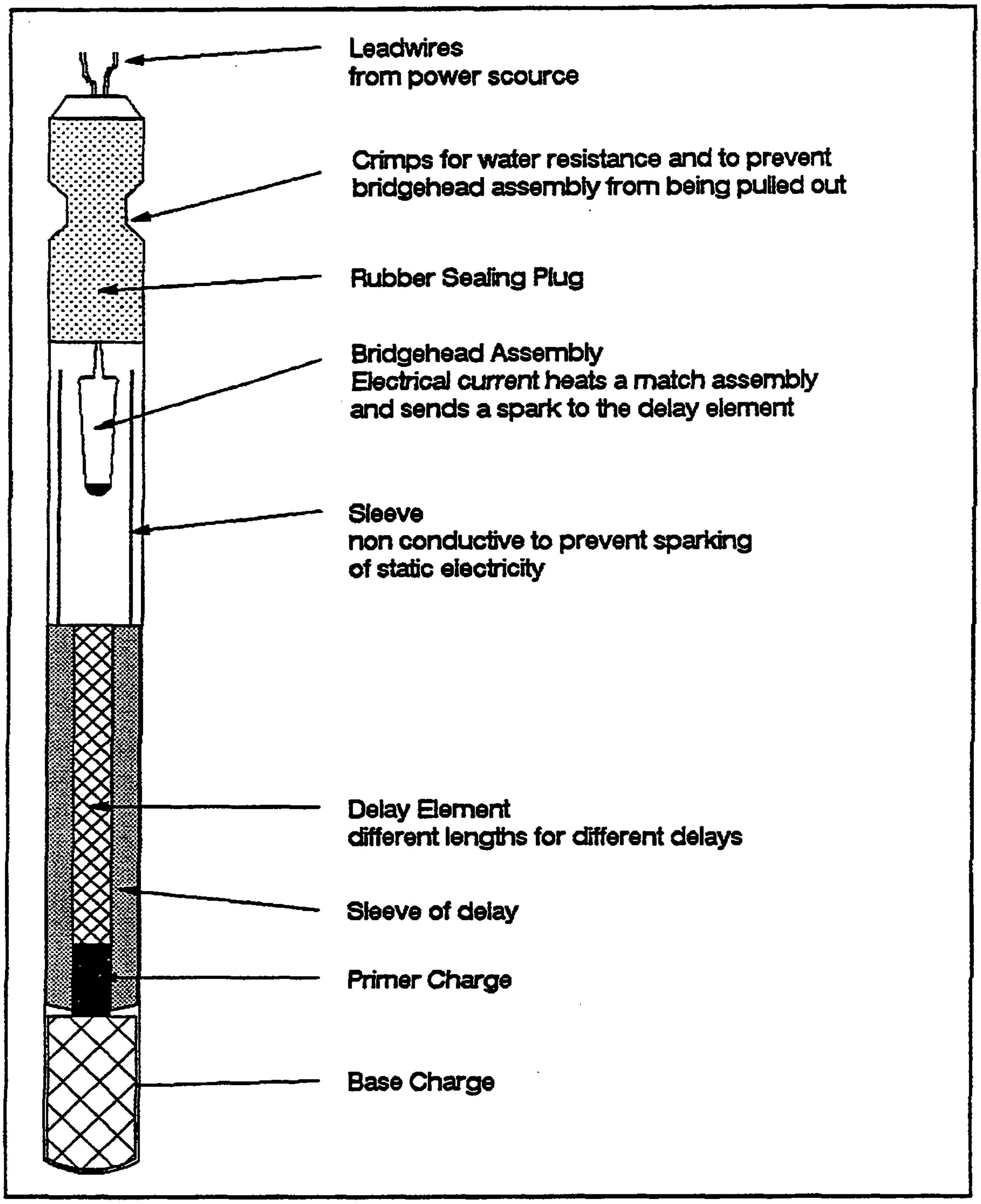
Electric and Nonel detonators are made in three different designs.

1) **IED** - instantaneous electric detonator. This has no delay element and is used, for example, to initiate detonating cord at the start of a blast.
2) **MS** - IT is the same basic construction as the IED but includes a delay element. The delay interval is measured in milliseconds (MS). Generally used for bench blasting in a quarry.
3) **LP** - The delay element is of a different composition which permits a long delay (LP) measured in seconds or parts of seconds.

5.1 Electric Detonators

An electric system uses electric power with an associated circuit to convey an impulse to electric detonators. These will fire and initiate an explosive charge. Inside the detonator the electrical energy is converted to heat by passing the firing current through a high resistance bridgewire. The heat energy ignites a pyrotechnic surrounding the bridgewire. The resulting flash ignites the delay element which then burns for a designated time and in turn ignites a base charge in the toe of the detonator, this initiates the primer. Figure 12 shows a section of an electric detonator. The electric blasting circuit consists of three elements:

Figure 12: Section of Detonator



5.1.1 Detonators

With lead wires connected in series or parallel or a combination of each.

5.1.2 Circuit Wiring

Connects the detonator circuit to the power source. Two wires of a resistance of about 1.5 ohm per 100 meters known as the blasting cable.

5.1.3 Power Source

Commonly called a shot exploder, to provide the electrical energy to the firing circuit.

5.2 Detonating Cord

This is a flexible tube containing a center core of high velocity explosive, usually PETN (pentaerythritol tetranitrate) that is used to, either, detonate explosives or transmit a detonation wave from cord to cord. The core of explosive is covered with combinations of materials to protect it from misuse or water and to give it strength. It is used in conjunction with relays to provide short firing delay intervals. Certain detonating cords can be used down large diameter holes as their tensile connected to a lower strength cord on surface which contain the relays. Detonating cord comes in reels of about 200 meters and can be cut to any length. A simple knot is enough to ensure propagation of the shock across a join.

5.3 Blasting Cap

Initiated by a safety fuse, it comprises an aluminum tube loaded with two charges. A base charge of a high explosive, PETN and a primer charge of lead azide. The primer charge changes the burning of the safety fuse into a detonation and initiates the base charge. It is used with igniter cord, which initiates the fuse, as an inaccurate delay system to detonate multiple small diameter hole blasts.

5.4 Nonel

In order to reduce the problems associated with electric delay systems, stray currents and earth leakage being two of the main ones, a system of non-electric (nonel) was invented. It uses a thin transparent plastic tube of 3 millimeters in diameter to transmit a low energy signal to a detonator at 2000 m/s. The tube contains a thin coating of reactive material on the inside which in itself will not detonate explosives. The signal is initiated by either a detonator or detonating cord. Various lengths of tube with a detonator are sold as a unit.

5.5 Delay Blasting

Millisecond delay blasting was introduced to the quarries many years ago. When blasting the rock movement time is very important, particularly in multiple row blasts. With one row of holes movement generated by blasting is directly away from the face, in an almost horizontal direction. As the number of rows increases so will the rock movement tend towards the vertical (flyrock). This is caused by the low velocity of the broken impeding the movement of that behind. The time requirement between rows to permit the rock to move in multi row blasts is 8 to 10 milliseconds per meter of drilled burden. For multiple row blasting the optimum delay is within the period that results in good fragmentation without the presence of cutoffs. Where the number of rows exceeds five or six an increase in the delay time is needed to successfully break the back rows without causing flyrock.

The true burden is dependent on both the drill and delay pattern selected. Initiation sequence affects the principal direction of rock movement. The best fragmentation is achieved when each charge is given enough time to detach the rock surrounding it before the next charge detonates. An additional free face is available to the next charge and the residual stresses in the rock are high enough to assist in the breaking by the subsequent charge.

The simplest type of delay blasting is to fire a single row of holes with delays between the holes. This system is applied to large diameter drillholes used in many quarries and results in better fragmentation than instantaneous blasts, where no delays are used.

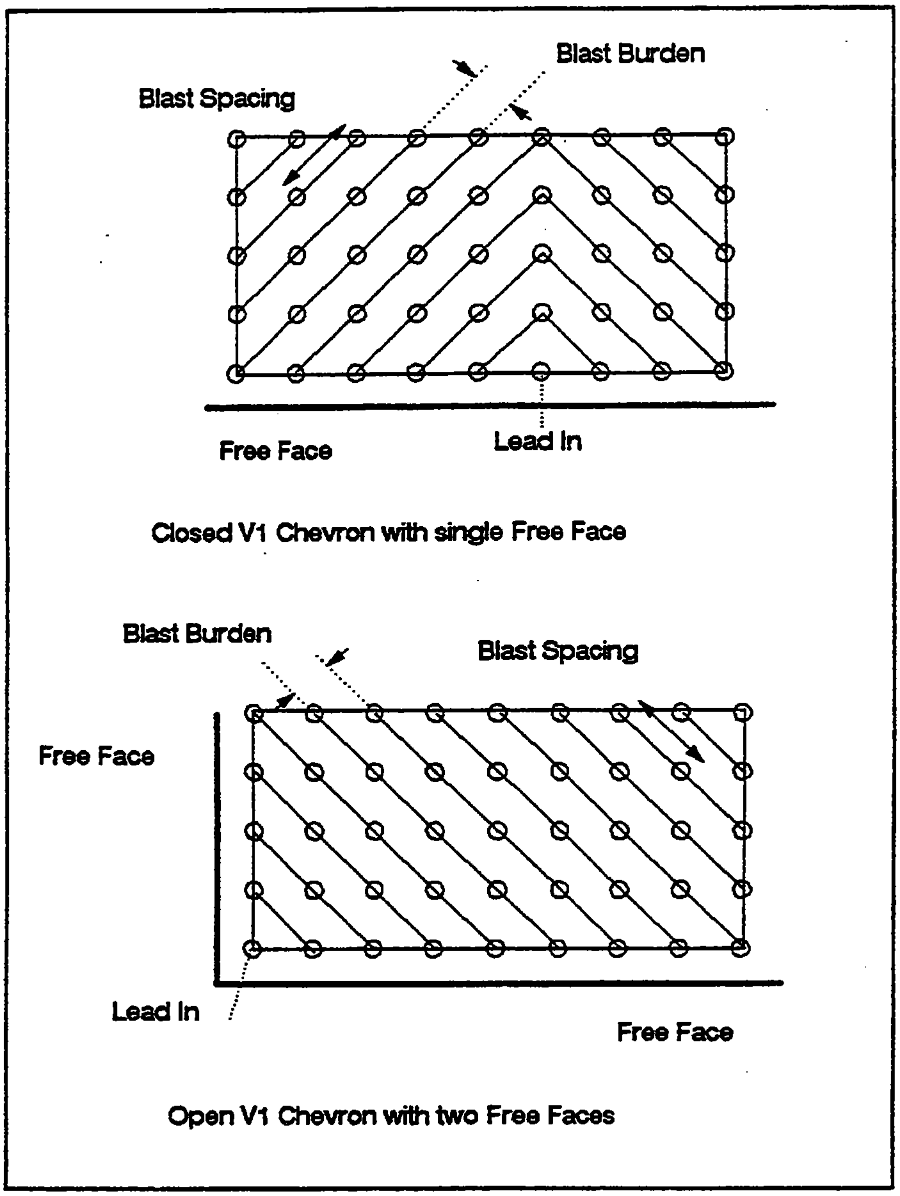
Blasting laws often limit the mass of explosive that can be detonated at one instant in time. This is particularly so where ground vibrations must be kept to a minimum. For example the proximity of quarries to built up areas.

5.6 Blasting Patterns

The length of most primary blasts, in relation to the number of rows, is such that initiation of a row will mean that a large number of holes will be detonated simultaneously. A 'V' pattern reduces this number as well as reducing the blasted burden of the holes. Change in firing direction will reduce the mass of explosive detonating at any one instant and therefore the vibration level. The type of pattern used will also regulate the position and height of the muckpile.

Figure 13 shows a 'V1' connection which can be either an open or closed chevron.

Figure 13: Timing Patterns



Blast Spacing to Burden ratio < 4 to 1

6. EFFECTS OF BLASTING

6.1 Fragmentation

The extent to which rock is broken into pieces by blasting.

Degree of fragmentation desired is dependent on the loading and crushing equipment and use of the product. In the economics of blasting cheaper means coarser fragmentation but this requires larger loading equipment to handle the oversize. Bigger machinery is designed for bigger tonnages not to load out oversize. Every quarry manager should consider the increased cost of drilling and blasting against the profitability of an increase in output obtained by an increase in fragmentation in order to obtain the lowest cost per unit for the whole operation. Many factors affect fragmentation. The major ones are discussed.

6.1.1 Terminology

'Fragmentation' is a general and highly subjective term in which 'good' does not necessarily mean 'small pieces'. Depending on the chief requirement of the operation and the final product management will focus on one of the following:

- Fines
- Oversize
- Mean Fragment Size

It is these terms that define fragmentation.

6.1.1.1 Fines

Rock particles of suitable chemical quality but too small for processing or sale. These represent an effective loss of reserve and production. Fines incur costs in production and additional costs in storage or disposal if not used. In a limestone quarry the fines can either be represented as increased capacity to the plant, they need no further reduction in size by the crusher, or as the overuse of explosive energy negating the use of the crusher.

6.1.1.2 Oversize

Rock particles too large to be handled by the available loading or crushing equipment. This causes delays in production while being moved out of the way and increased maintenance costs to loading equipment not designed to handle them. They generally increase working costs, the majority have to be re-broken, and reduce the safety of both workers and equipment. It is normal practice for management to measure the blast purely on the percentage oversize contained in the muckpile and to disregard the percentage fines.

6.1.1.3 Mean Size

Defined as the mesh size through which 50 % of the muckpile can pass. While not as obviously critical as the above it provides a meaningful guide to ease of digging. Being at the midpoint of the size range 'mean size' is not as sensitive to small variations in the 'oversize'. For example a 100 % increase in boulder count may represent less than a 1 % increase in mean size.

6.1.2 Quality of Explosives

The product of strength per unit weight and charging density, known as strength per unit volume, determines the effectiveness of different explosives in breaking rock.

6.1.3 Rock Characteristics

In jointed rocks the correct direction of blasting is important. In hard, solid and slightly fractured rock the extent of the crushed zone around the blasthole is dependent on the charge per hole length.

6.1.4 Blasthole Loading

Proper fragmentation occurs when there is enough force in the compression wave to travel to the free face and back. The quantity of explosives is enough to fragment the rock. If not enough stemming is placed in the collar of the blastholes then, on detonation, gasses will escape from the holes reducing the efficiency of the explosives.

6.1.5 Drilling Accuracy

If drilling accuracy is suspect the planned drilling parameters will have to be decreased to maintain explosive efficiency.

6.1.6 Timing and Pattern

Better breaking has been found when using a delay interval, in milliseconds, of about ten times the drilled burden in meters. The interval should be chosen after taking into account the burden and spacing and the number of holes to be blasted.

The use of 'V' patterns will somewhat reduce block size by causing collisions of the rocks during heave. It also results in easier to load muckpiles. In any blast the stemming area produces the most oversize as there is usually a lower distribution of explosives than in the rest of the blast. To improve fragmentation the quantity of explosive in the collar area can be increased in a number of ways. The column charge can simply be increased. Smaller diameter and shorter length holes can be drilled in between the main holes and charged with a small quantity of explosives. A separate charge can be introduced into the stemming, known as decking.

6.2 Muckpile

This is defined as a pile of blasted rock that is to be loaded for removal. The 'throw' or movement of the rock from the blast increases as the specific charge increases and may be controlled by varying it. As blasting proceeds in multi row conditions the muckpile in front of the face will gradually lie closer to it. The shape of the muckpile is influenced by several factors.

6.2.1 Drill Hole Angle

An inclined hole will project the rock further than a vertical hole resulting in a flatter outline of the blasted rock.

6.2.2 Surface Timing

A closed 'V1' pattern will result in a steeper muckpile. As the two sides of the 'V' detonate the rocks collide in the air and drop. Open 'V' pattern or lineblast will give a flatter outline to the resulting rock mass.

Diminishing the burden or increasing the hole size will result in the rock moving further and giving a flatter muckpile. The type of loader used in the quarry will determine the most efficient muckpile outline. A front end loader will find it easier to load out a flatter muckpile and a face shovel a steeper one.

6.2.3 Free Face

It is assumed that blasting cannot be effective unless a free face exists towards which movement can take place. This is not so, the free face improves the efficiency of the breaking but does not prevent fragmentation from occurring. Choked faces inhibit movement and cost more in explosives consumption, but probably do not significantly affect the fragmentation. In solid ground, without a free face, high powder factors are necessary for good fragmentation. A degree of movement is enabled through the porosity of the rock and voids left by previously fired holes. The practice is not common as other factors mitigate against it, namely ground vibrations, airblast and flyrock.

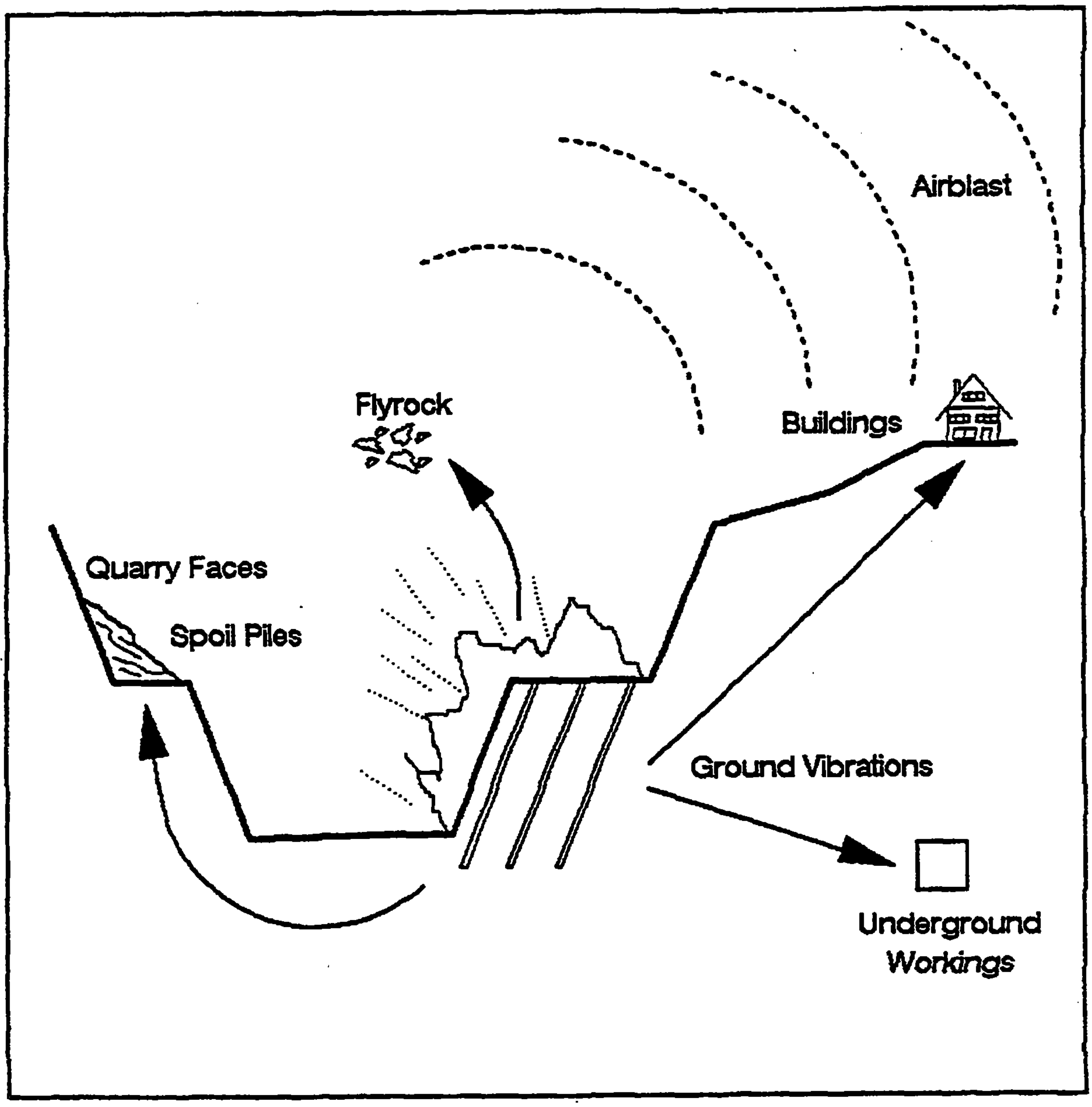
6.2.4 Fragmentation Analysis

The only sure method of obtaining a muckpile fragmentation analysis is to screen the whole muckpile. Any method developed would also depend on this for an absolute correlation. This is clearly impossible and all methods, therefore, have an unknown degree of error. This can be minimized by working on comparisons, the error then being constant. With time and experience the error can be assessed and almost eliminated.

6.3 Ground Vibrations

The trend towards larger holes and bigger blasts and an increased population has highlighted the problems of ground vibrations. Areas of concern when blasting takes place are shown in figure 14. Damage due to old age or settlement of a building is very hard to distinguish from blast damage. The main criteria is to reduce the level of vibration to such an extent that complaints are minimal. At the same time good public relations will ensure that people living locally will feel reasonably well disposed to the quarry.

Figure 14: Areas of Concern



When an explosive detonates in a hole it generates an intense stress wave on both transverse and longitudinal wave motions. This motion crushes the rock around the hole up to about one drillhole radius and permanently distorts and cracks it for several more. Most of the energy is spent on shattering the rock but because of the imperfect nature of the explosive, some of the energy is transmitted through the ground as vibration in the form of elastic compression waves. This represents a transfer of energy from one point in the rock to another. Initially there must be some displacement of the rock as certain forces act to displace it from its equilibrium position and introduce new energy to the system.

6.3.1 Source of Ground Vibrations

If the rock does not exhibit an elastic response, energy is absorbed by it and only dampened waves come from the blast area. If elastic response is exhibited then the action of the blast causes nearby portions of the rock to oscillate about their rest positions, similar to a spring. Oscillatory conditions are set up and the disturbance is transmitted from one element to the next as from the rock to a building. During wave motion there is no bulk movement of matter. The transmission of waves is affected by distance from their source. The rock through which they travel is never perfectly homogeneous, it contains deformities such as joints, bedding planes and different rock types.

Total energy of a ground motion wave generated varies directly as to the mass of the charge detonated. As it travels outward from the inception point the volume of rock affected by the compression wave increases and the peak ground motions decrease. The ground motion wave of a column charge of explosives, where the length to diameter ratio is greater than 6, takes the form of an expanding cylinder. The volume of this compression cylinder varies as the square of its radius. Thus the peak level of the ground motion is inversely proportional to the square of the distance from the blast. In the majority of quarries the detonation of a borehole takes the form of a cylindrical blast. For example a 104 mm hole has to have a minimum single charge length of 0,625 meter for it to be a cylindrical charge.

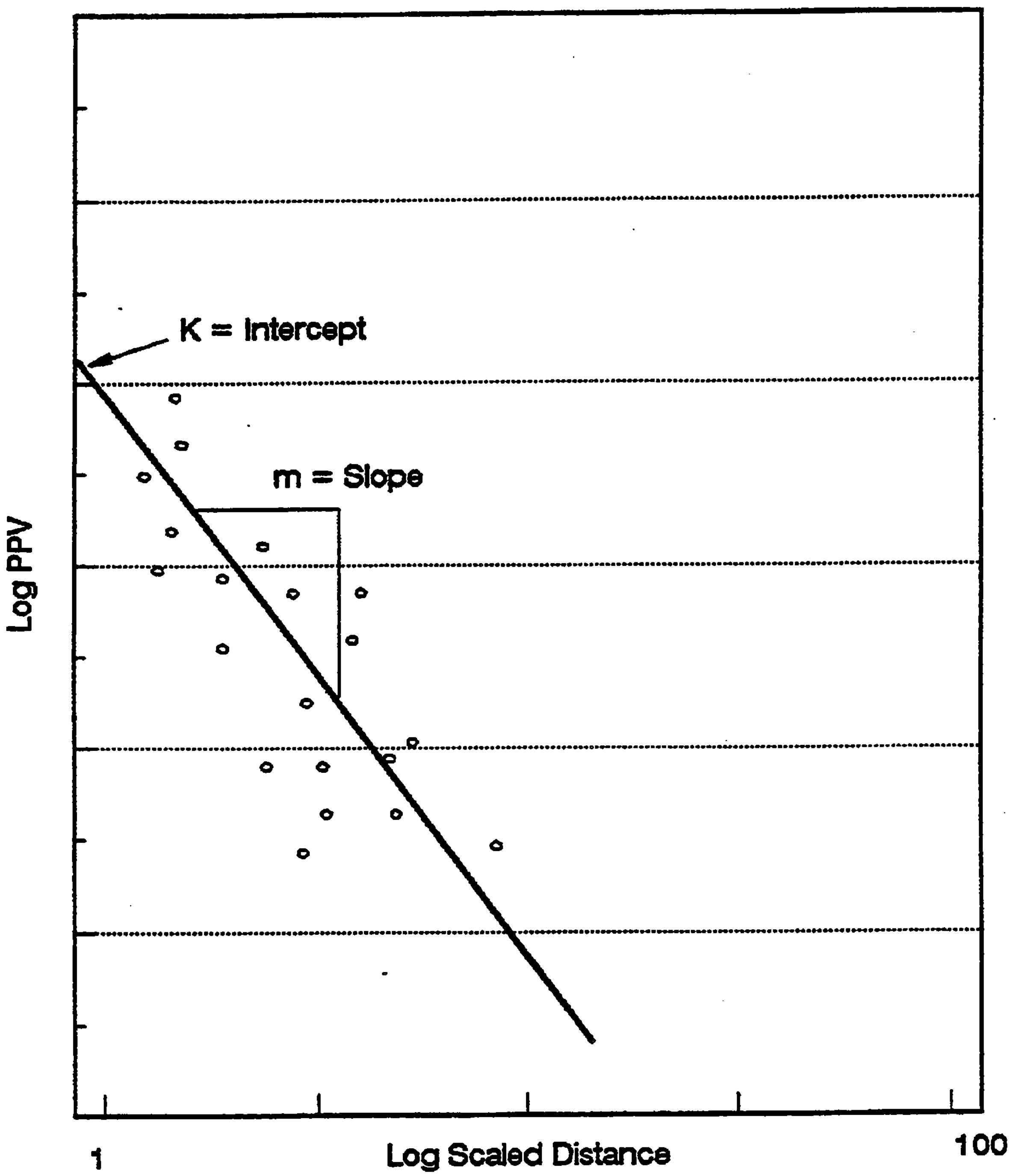
6.3.2 Defining Peak Particle Velocity

The empirical scaling formula relating peak particle velocity (PPV) to scaled distance has been developed from actual field results. Scaled distance, d/(W^1/2), combines the effect of total charge weight per delay, W, on the initial shock level with increasing distance, d, from the blast. The formula contains two site factors, K and m, which allow for the local influence of the rock on the rate of peak particle attenuation. Geometric progression is included in the slope exponential (m) in the following equation:

V	=	K(d/(W^1/2)^-m
where V	=	Maximum PPV (mm/sec)
d	=	Distance from blast (m)
W	=	Mass of explosives per delay (kg)
K & m	=	Slope of graph
d/(W^1/2)	=	Scaled distance for a cylindrical charge

Site factors are determined from the logarithmic plot of PPV verses scaled distance. The graph is drawn and the best line representing the data is inserted, see figure 15.

Figure 15: Qualitative Graph of Ground Vibration



Showing the Empirical Site Factors

Work done by the US Bureau of Mines implied that the best damage indicator is PPV as a function of frequency below 40Hz. A number of studies have correlated levels of PPV with resulting damage and an example of the levels is shown in table 3.

Table 3 Ground vibration levels

PPV (mm/s)	Effect
300	Rock falls in unlined tunnels
200	50 % probability of major plaster damage
135	50 % probability of minor plaster damage
80	Threshold of damage
50	Limit of safe blasting > 12 Hz by USBM
25	Limit of safe blasting < 12 Hz by USBM

The intensity of seismic waves that can be tolerated by structures varies as to the method of construction. A steel reinforced building can withstand a higher level of motion than a privately owned house built of brick. Plaster is commonly used on the inside walls of houses. It is relatively weak when compared to other building materials and is used as the basis for damage criteria.

6.3.3 Techniques to reduce Vibration Levels

To minimize the level of ground vibrations it is easy enough to reduce the charge per delay. This however may not be practicable as it implies reduced production. Several other steps may be taken to reduce levels of vibration:

6.3.3.1 Blast Design

This should give maximum relief of burden. Internal free faces in the blast can reflect compressional waves.

6.3.3.2 Powder factor

An excessive powder factor can increase ground vibrations and may cause excessive throw of the muckpile. Vibration levels can also be increased by an insufficient powder factor. It delays and reduces the effect of rarefaction waves reflected from free faces.

6.3.3.3 Spacing to Burden Ratio

To be greater than one. Overburdening of holes can cause them not to break out to the free face. This results in excessive explosive energy used as shock waves through the rock.

6.3.3.4 Accurate Drilling

Poor drilling can over or under burden the holes with the same results as for poor spacing to burden ratios.

6.3.3.5 Sub Drill

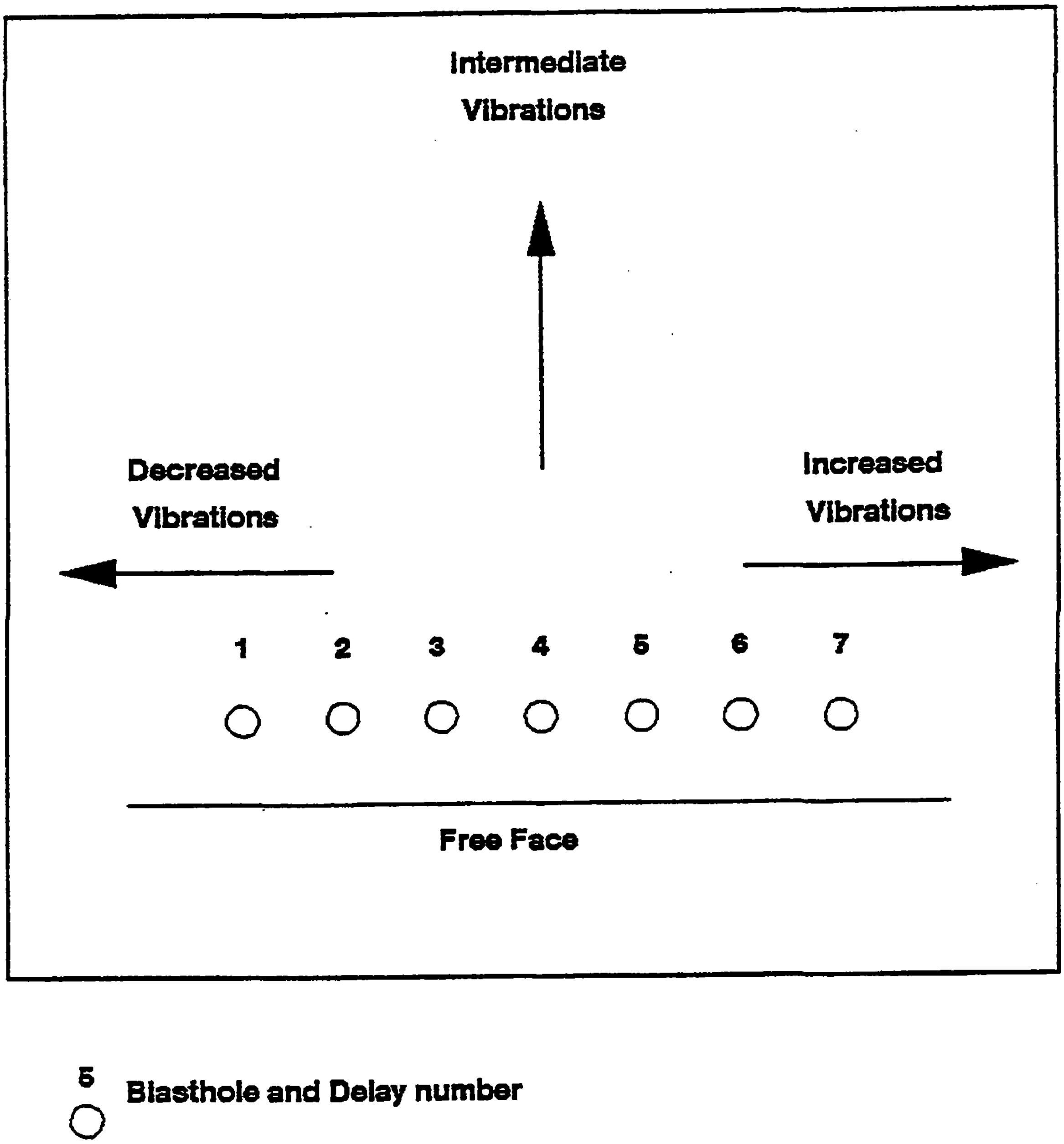
The toe of a drill hole is the most confined area and has the most difficulty breaking out. Over drilling increases the amount of sub drill and therefore the vibration levels.

6.3.3.6 Adequate and Accurate Delays

A long delay period between holes will guarantee that they do not detonate simultaneously. Shock waves can be reinforced by waves from subsequent holes in a line detonating if the delay interval is insufficient, see figure 16.

Too few delays in a blast results in a high mass per delay. Studies have shown that millisecond delays in commercial detonators are not very accurate. This can result in close timing or, in extreme cases, an overlap.

Figure 16: Delay Sequence and PPV



6.4 Airblast

The compressional wave in air, either from unconfined explosives or by indirect action of a confining material subject to explosive loading, is known as airblast. Noise is that portion of the spectrum in the range 20 to 20 000 Hz. Airblast at levels below 20 Hz is known as concussion. Large burdens, typically found with large diameter boreholes, means that the airblast contains a considerable amount of energy at frequencies below 20 Hz. Low frequencies can damage structures directly, but more often causes higher frequency vibrations in windows and doors.

Table 4 Airblast

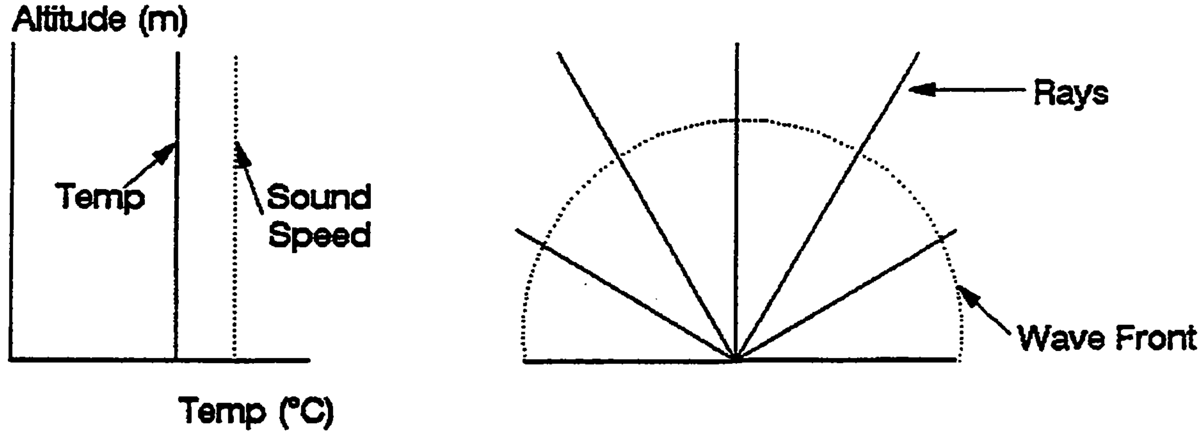
Noise (dB1)	Effect
180	Conventional structures break
170	Most windows break
150	House windows may break
140	Large plate glass windows may break
136	Interim limit USBM damage to hearing
80 - 100	Loud Radio / TV

6.4.1 Atmospheric Conditions

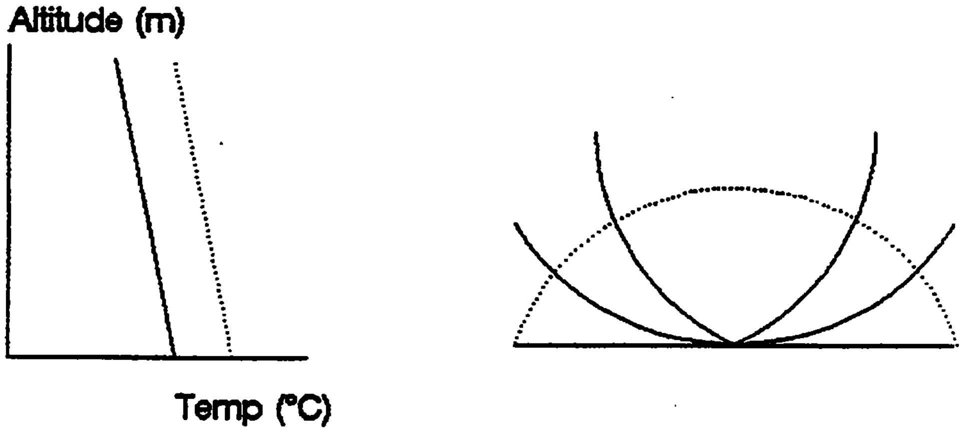
Atmospheric conditions may affect the intensity of the noise at a distance from the site of the blast. The speed of sound in air varies at different altitudes and temperatures. In addition the wind speed will affect the distance that sound waves carry. Normally the air temperature decreases with altitude, the adiabatic lapse rate, at a rate of about 2.0 degrees centigrade for 300 meters increase in altitude. An inversion, a decrease in temperature with altitude, will cause noise to be reflected back to earth causing excessive noise levels to be heard in unexpected areas.

As the time of day can determine weather conditions, inversions occurring more often in the morning, blasting times should take note of this fact to minimize possible complaints. The effect of wind on noise levels is greater during the cold months because of the higher wind speeds. This helps to prevent inversions occurring. Figure 17 sketches the effect of varying temperatures with altitude wind on noise levels.

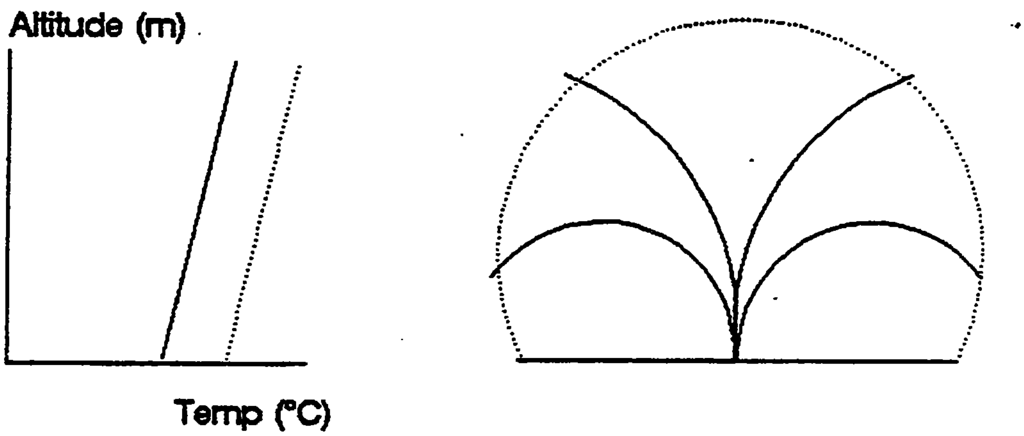
Figure 17: Airblast - Effects of Temperature and Altitude



No Problems



No Problems



Inversion - Potental Problems

6.4.2 Minimizing Airblast

Minimizing airblast not only depends on the correct weather, a clear day with light wind, but also:

6.4.2.1 Use of Adequate Stemming

Confine the blast stemming blown from the hole results in higher noise levels.

6.4.2.2 Secondary Blasting

Not using mudblasts, breaking rocks with a lay-on charge which is covered with mud.

6.4.2.3 Sequential Blasting

Sequence of blast must proceed in a proper order. Out of sequence shots cause excessive airblast.

6.4.2.4 Time of Day

Scheduling of blasting operations when people are busy. Rush hour generates a high level of noise which will cover blasting noises.

6.5 Flyrock

This is the undesirable throw of rock from a blast. Rocks have been thrown many hundreds of meters and caused both material and bodily damage. It is probably the biggest cause of personnel injury and property damage in the whole blasting operation.

Flyrock is caused by a number of factors:

1) Overloading of holes or insufficient burden
2) Poor quality stemming
3) Incorrect timing of surface delays

Regardless of the care taken with a blast flyrock is always possible and every precaution taken to prevent damage. All blasting personnel must take cover when the blast is detonated. The direction of the blast must be away from any structure which could be damaged.

A quarry situated close to built up areas has to prevent flyrock at all costs. Good blast design is the best method of minimizing this hazard.

7. LOADING AND MUCKING

The methods of loading and hauling in quarries can be divided into four categories:

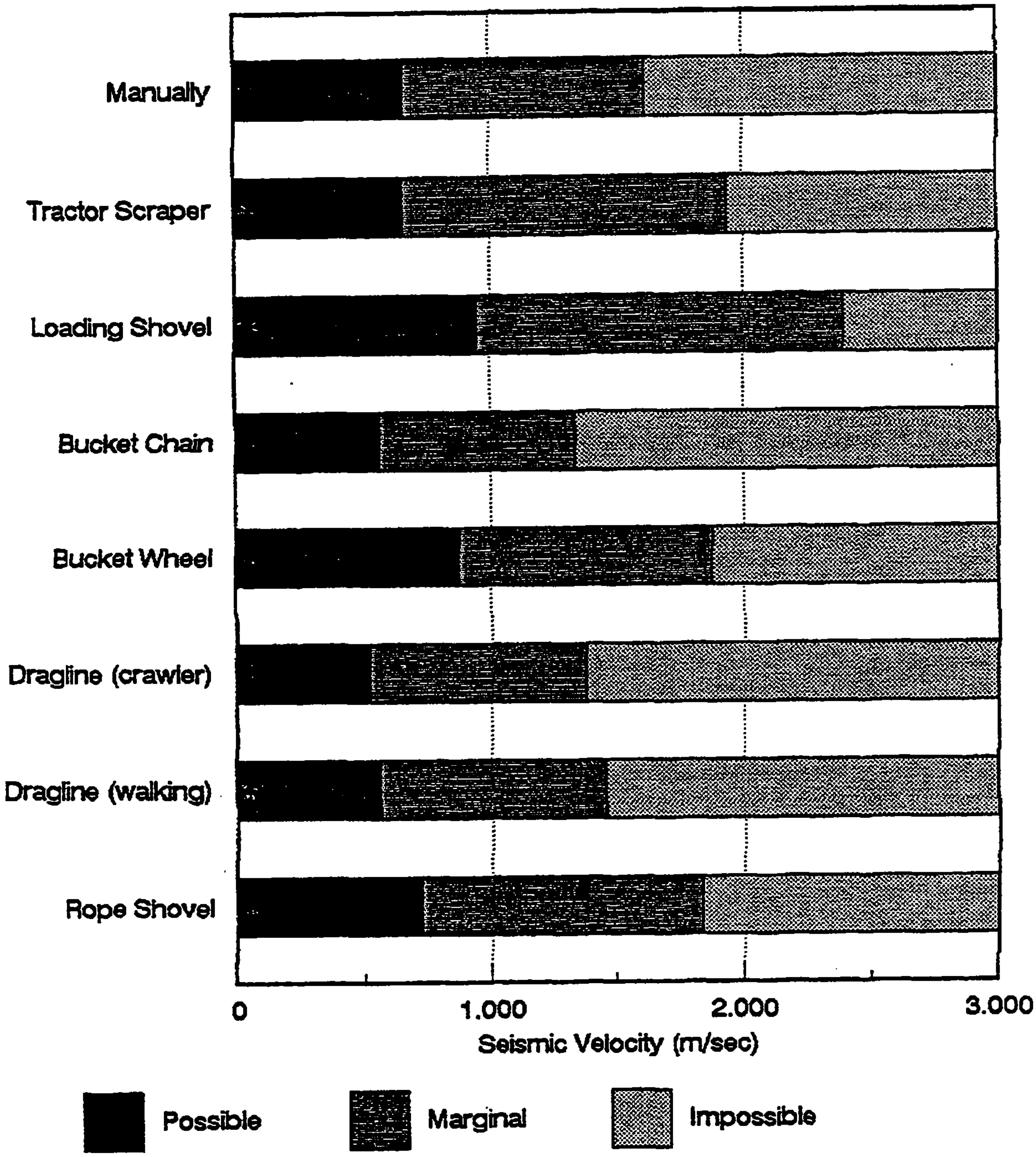
1) Continuous mining systems such as the bucket wheel excavator.
2) Other non-blasting systems such as ripping and bulldozing or scraping.
3) Haul road, using loaders and trucks and a ramp to exit the quarry.
4) In pit crushing in which a loader or face shovel supplies a mobile crusher.

Equipment to load and transport rock is dependent on the hardness of the rock to be quarried. Soft rock or overburden can be removed without the need for blasting by ripping and dozing with specialized machinery. There are both discontinuous and continuous systems for the removal of rock without blasting.

7.1 Selection of Equipment

Initially the quality and quantity of any deposit must be established to enable a decision to be made as to the mode of operation. Seismic wave prospecting can be used to discover the hardness and extent of the deposit. This data is compared to previous data obtained in the exploration of other deposits. For many commonly found rocks a range of rippability values in terms of their seismic wave velocities have been tabled with a fair degree of accuracy. Figure 18 shows a comparison of wave velocity ranges over which different loading equipment can be expected to work.

Figure 18: Excavation Method for Seismic Velocity without any blasting to loosen rock



Depending on the planned tonnage to be excavated per year and the consistency of the deposit, both physical and chemical, over large areas determines the equipment to be purchased. Large tonnages of relatively soft material with a shallow deposit favor a continuous mining system. Smaller production levels with varying quality of limestone favor a more mobile system such as a truck and shovel.

The final selection must take into account overall economics and will be based on balancing the following against each other:

1) Capital Cost
2) Technical Suitability
3) Maintenance and Repair
4) Manufacturers Acceptability

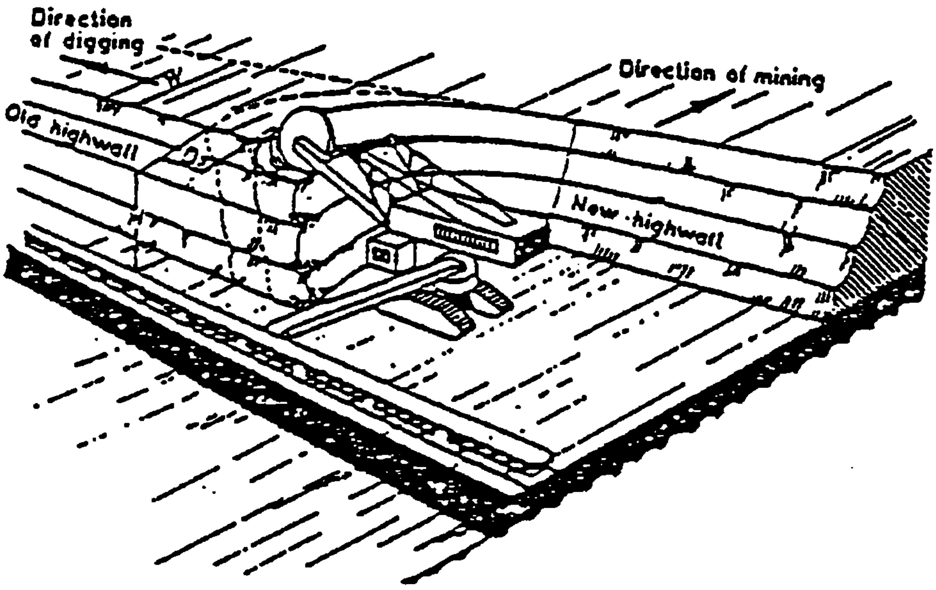
7.2 Non-Explosive Mining

There are several systems of exploitation that do not require the use of explosives. They are generally confined to quarries that have soft rock with a uniform quality over large volumes with few inconsistencies permitting a continuos mining machine to advance in a systematic manner with little or no manoeuvring. Chalk or Marly Limestone are good examples of these softer rocks that are excavated as raw materials in the manufacture of cement. Soft rock permits the material to be exploited without it first having to be drilled and blasted. The rapid advance of mining technology has developed machines that have applications in harder rocks but to be cost effective require a high tonnage output. Capital cost of most continuos mining machines are too high to warrant their use in what is essentially a small tonnage per year operation at a cement plant.

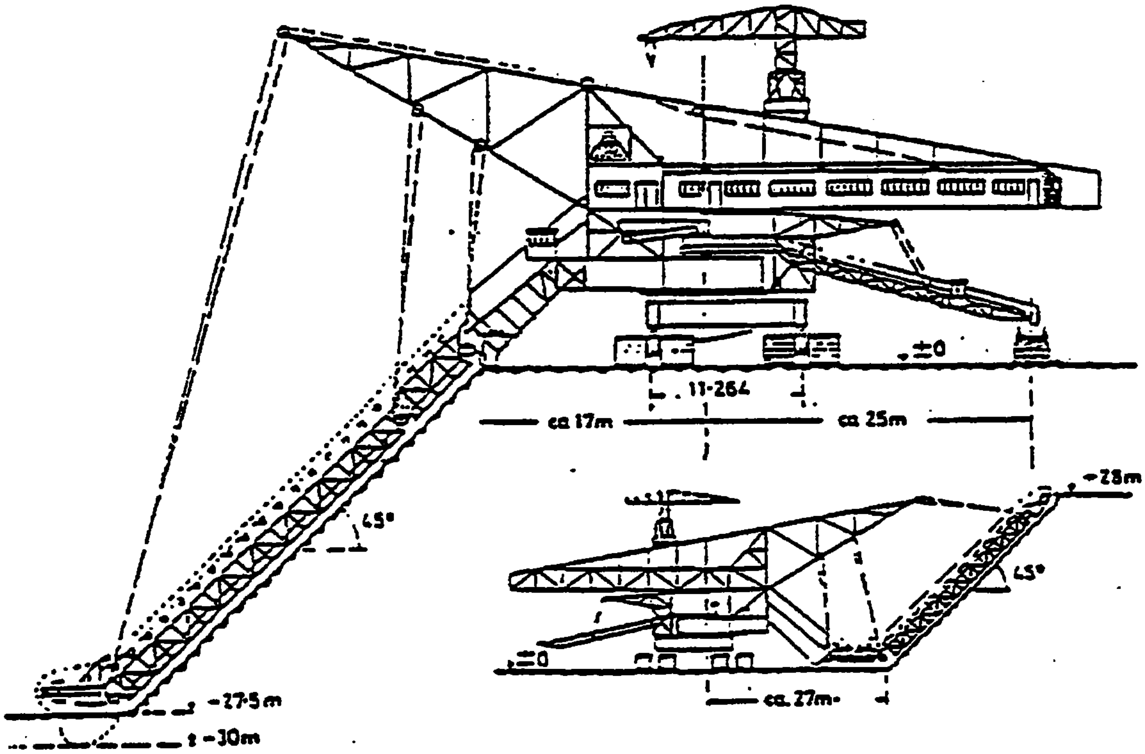
7.2.1 Continuos Mining Systems

Excavation of raw material can be accomplished by the use of a Bucket Wheel Excavator (BWE) or Bucket Chain Excavator (BCE), examples are sketched in figure 19, or a newly developed Surface Miner. If considered as an alternative to conventional exploitation methods the capacity of the plant has to be sufficient to accept the high tonnages involved, some BWE's will move 100'000 mts per day. There also has to be a continuos system of removing material to the plant if the machine is to have a high utilisation time. This usually takes the form of a conveyor belt. Any alternative, such as truck or rail systems, can quickly result in the excavator waiting for loading equipment. Non-continuos systems of transport require detailed planning for them to be used to their fullest and minimise downtime of such a machine. Optimum operation is normally achieved by building an excavating machine and a complete transport system specifically for the required application.

Figure 19: Continuous Mining System



Bucket Wheel Excavator



Bucket Chain Excavator

The advantage of a continuos mining system are that it reduces:

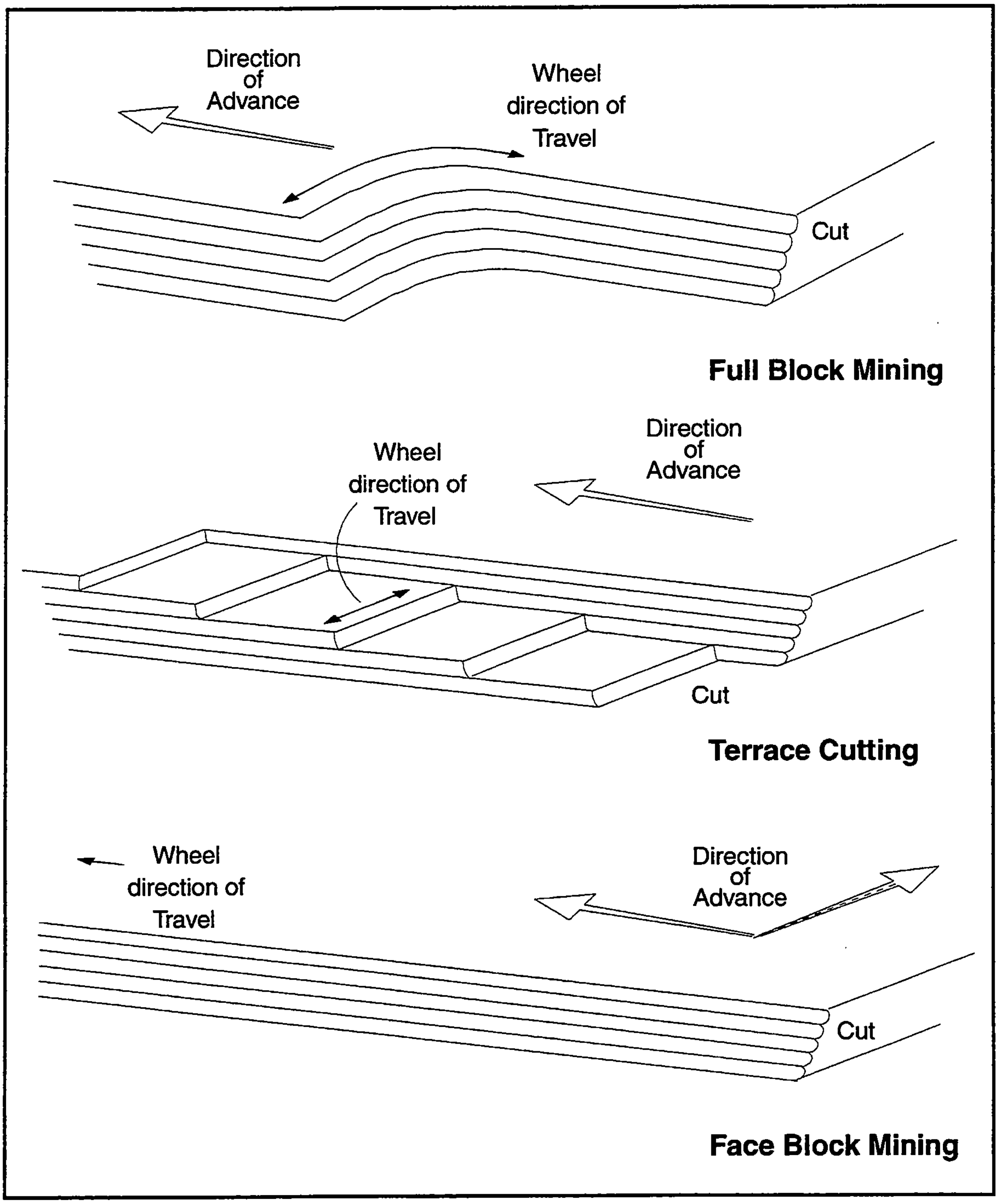
- dependence on diesel fuel, a primary fact if this has to be imported.
- labour complements; although it may be necessary to have a fleet of support vehicles such as bulldozers, personnel carriers and cranes to service the continuos miner
- noise, dust and air pollution in the operation of winning rock.
- haul road maintenance
- power consumption

Development of light weight machines, for example, tyre mounted bucket wheel excavators or surface miners which are considerably cheaper to purchase and have lower capabilities has made the use of these machines in our industry in the future more likely.

7.2.1.1 Bucket Wheel Excavator (BWE)

Methods of BWE operation are described according to the position of the machine and how it cuts the quarry face. Figure 20 shows typical methods of excavation with a BWE. In each method there are two alternative cutting techniques, horizontal, or terrace cutting and vertical, or drop cutting These are used in the selective mining of specific layers for reasons such as quality control.

Figure 20: Excavation Methods with a Bucket Wheel Excavator



Originally these excavators were designed to move easy digging materials such as sand, clay and loam. Today they are principally found in the lignite fields of Germany and Australia moving large quantities of overburden. However a newer generation of BWE has incorporated modifications to its design giving better operating parameters in the digging of harder materials. It is this type of machine that may be considered for use in the cement industry of the future. Although in extreme cases blasting can be considered an assist to digging of rock with these machines it should not be considered as a normal practice.

The BWE has little operating flexibility and a detailed study should be conducted before purchasing a machine. Included in this should be detailed quarry layouts based on exploration results and the geology of the deposit. Choice of machine in terms of output and maximum vertical digging depth has to be carefully considered. As a rule the vertical reach of the BWE should equal the final quarry depth.

Large BWEs can compare favourable with a hydraulic or electric shovel in the following areas:

- instantaneous power demand
- weight to output ratio
- power consumption
- wider operating benches and more stable quarry slopes
- close control in selective mining; albeit over large areas
- easier and cheaper rehabilitation of overburden
- an ability to exploit material both above and below the working floor

Selection of a machine for a particular production rate can be more complex than for other types of excavator. Machines of apparently equal capacity can vary in profile dimensions, weight, horsepower etc. A complete system may include a belt wagon, hopper car, mobile stacker and a cable reel car. With all these components the overall availability of the operation can easily fall below 50% unless the operation is carefully planned.

Digging height and cut width will be proportional to the boom length. Efficiencies will increase with greater digging and cut heights but have to be evaluated on the basis of the operational plan and the total economics of the operation given the increased cost of the equipment. In some cases it will be more efficient to use two smaller machines than one large; particularly if selective mining is considered or the stratagraphic layer to be mined is thin. Two BWEs will permit some production if one machine breaks down.

Additional considerations for the selection of a BWE are:

- type of bucket
- tooth design
- transfer feeder for material from the wheel to the belt
- suspension systems
- wheel drives
- boom type

A long depreciation period is necessary for a BWE to be economical. Larger machines can be considered to have a life span of more than 30 years. Due to its tremendous digging capacity and its high initial cost to achieve the necessary low cost per tonne necessary the machine must operate around the clock with only stops for planned maintenance. It is not uncommon for such a machine to have planned weekly operating hours of 132.

Operating costs are based on two rules :

- digging wheel costs represent 85 to 95% of the total excavator maintenance costs
- total repair costs amount to 5 to 8% of the total excavator costs per year.

Features that greatly increase the cost of a unit are:

- Efficiency Increases in the digging ladder
- Capability of digging below the working floor
- Increases in the normal wheel and bucket size

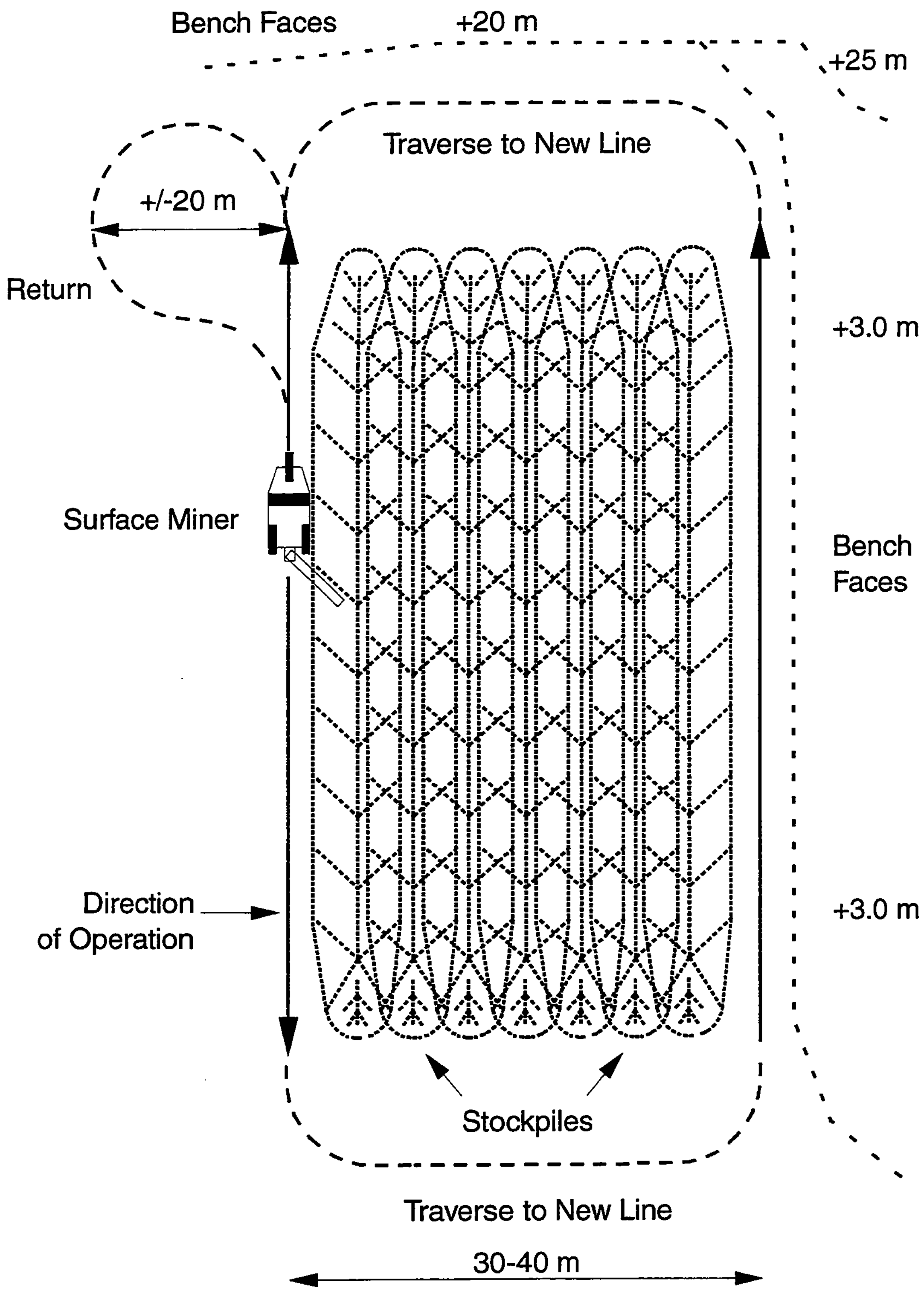
7.2.1.2 Surface Miners

This a relatively new inovation in the field of continuos miners for use in limestone quarries. It was developed from road repair machines for the quick repair of asphalt road surfaces. Machines consist of crawler mechanism for driving and steering mounted in a frame with a mounted drum of between 1.0 and 1.8 metres diameter and about 3 to 4 metres width fitted with rows of teeth or picks. The drum on quarry production models is often centrally mounted. This position of the drum ensures that the machine's weight maintains pressure on the drum while it digs into the rock. Material is removed by means of a short conveyor belt mounted behind the drum. This will either feed cut rock to a conveyor or load it into a haul truck. The cutting drum rotates in an up-cutting direction, the picks chip at the rock as the drum rotates, and has spirally welded deflection segments attached to ensure that the cut material is directed to the extraction conveyor mounted behind the drum. The rows of picks are tungsten carbide tipped to maximise life and they can dig a "trench" of between 250 mm and 600 mm depth. Broken material is transferred to an on-board conveyor and removed.

Production tonnage's depend very much on hardness of rock and the mass of the machine. The quantities of broken rock produced vary from a low of 300 mts per hour with a small machine up to 800 mts per hour with the largest at present manufactured. Typical product of this type of machine has a high degree of fines, -5.0 mm, with a maximum size of 120 mm. However this can be somewhat adjusted by altering the number of picks and the rotational speed of the drum. One advantage of these machines is that the exploitation and primary crushing are a single process; it does not require a separate primary crusher and in certain cases no crusher is needed in the process. However in feasibility studies a secondary crusher has always been included and if feed material to the raw mill has to be – 25 mm then a secondary and tertiary crusher will be necessary to reduce material to an acceptable size for feed to a ball mill.

Careful planning of the quarry operation is necessary to realise the full economics of the operation. The machine is slow; speeds of about 2.0 ks per hour while digging rock and 6.0ks when moving to and from site can be expected. The quarry floor should be reasonably flat and level over a relatively large area to accommodate a machine. As the exploitation operation is a continuos one material is preferably removed from site by a conveyor belt. However haul trucks can be used. These run alongside the miner in the same direction and are loaded on the move by the discharge conveyor on the surface miner. Figure 21 shows the building of stockpiles of limestone in a quarry where neither haul trucks or a conveyor belt were available. The stockpiles were subsequently loaded using a front end loader.

Figure 21: An Example of Surface Miner Operation



The advantages of this type of machine are:

- labour costs are low, only a single person is required to operate it
- eliminates blasting and minimises dust, the machine is quiet in operation.
- produces ready crushed material

Disadvantages are:

- complex machine requiring considerable maintenance
- as a single production unit breakdowns result in a stoppage of raw material supply.
- requires detailed planning to operate economically and with a high percentage utilisation
- not really suited to haul truck loading

A Surface Miner is considerably cheaper than a BWE but can still cost in the region of USD3.00 million. To maintain some production it would be necessary to purchase two machines if this form of mining was considered.

7.2.2 <u>Semi-Continuos</u>

Other non-explosive systems of winning material find more favour due to the ability of such a system to relocate and their relatively low capital cost per unit. Examples of these are the use of a shovel to load material directly into a haul truck or the use of a ripper on a bulldozer to loosen the rock and a load haul system to move material to the crusher. This enables a plant to have the operational flexibility provided by the use of multiple mobile equipment while not having to invest in new mining systems.

7.2.2.1 Excavation by Face Shovel

The principle reason for the preference of front end loaders in our industry is that they are versatile and highly mobile. They do not have the break-out and crowd forces of a face shovel but this is compensated for by blasting techniques sympathetic to the loading characteristics of the machine. However in certain cases hydraulic face shovels are used in a quarry. This occurs most often where the drill and blast operation has had to be terminated because of environmental considerations and the rock is too hard to be dug with a FEL. In addition manpower costs limit the number of units that can be purchased precluding ripping and dozing.

Material is broken out of the quarry face by a hydraulic face shovel. As the limiting criteria in such a case is breakout and crowd force these machines are often large and have a loading capacity far in excess of the plant requirement. Because of their relative immobility it is the norm to have several of these machines on-site. One such quarry operated four shovels, one on overburden, and three on raw material production; each in a different layer. Material is loaded to haul trucks in the normal fashion.

7.2.2.2 Dozing and Ripping

Material is excavated by a bulldozer pulling either a single or double "pick" at the rear of the machine. These "picks" dig into the ground and are pulled along by the bulldozer tearing out the rock to a maximum depth of about 75 cms. Subsequently the bulldozer pushes the broken rock into piles where it is loaded and transported to the plant. If the rock is soft enough not to require ripping the bulldozer will simply push it into piles with the use of the dozer blade.

The selection of the most suitable equipment for a particular quarry is important as not every machine will be capable of doing the job efficiently. It may be more cost effective to use blasting to loosen the rock even though it may be soft enough to rip or scrape. Selection of machinery for the ripping or dozing operation depends upon:

1) Down pressure available at the tip. The ripper must be able to maintain penetration of the rock.
2) Flywheel horsepower. Power to advance the tip.
3) Gross weight of the machine. This determines if sufficient traction horsepower is available to permit full drawbar pull.

Various combinations of ripping and dozing are in use, the ideal case being a one pass system. In some situations the dozer is used to physically push the ripper where the rock conditions are hard.

7.2.2.3 Scraping

Scraping involves both the loading and transporting of the rock in one machine. Conditions rarely exist where this system can be used as the rock must be soft or loose. These machines are used in conjunction with rippers to excavate material loosened by them. Scrapers can be used in overburden removal where it is soft.

7.3 Haul Road

This involves building ramps up or down the side of quarries and using dump trucks and loaders to haul rock from the blasted area to the primary crusher, usually situated outside the quarry. It is a very versatile method of operation and is suited to any terrain, from flat ground to mountainous, and any size quarry. This method of moving rock to the crusher is the most common in small to medium quarries. The trucks will range in size from 20 to 80 tons. Loaders range in bucket capacities from 3 to 13 cubic meters, based on the specific gravity of the rock to be loaded.

7.3.1 Loaders

Four types are commonly used in quarries:

- Electric Shovel
- Wheel Loader
- Hydraulic Excavator
- Tracked Loader

Electric mining shovels have been in use for many years and because they are simple machines tend to continue working efficiently. Their main disadvantages are their large initial cost and, if electric, lack of maneuverability. They have now nearly disappeared from the average sized quarries superseded by hydraulic loaders and shovels.

The advantage of wheel loaders is their high mobility and with recent advances in technology in respect to size and digability they can equal or exceed the excavating rate of face shovels. Front loading hydraulic excavators are a recent development. With their extremely short cycle time and relatively high power they are becoming more popular. The capacity of these loaders is quickly reduced with increasing distance between loading point, at the muckpile, and delivery to a truck. The optimum is 10 to 20 meters and for any distance greater than 120 meters the capacity of the loader is almost halved.

The most impressive feature of the hydraulic excavator is the high mobility of the bucket and the high breakout force which can be transmitted to the bucket. Because of its mobility it can be employed at a variety of locations in the quarry. A further advantage is that a hydraulic excavator will load at less cost per ton than a wheel loader. They allow for the careful, thus reducing spillage, and quick loading of trucks thus achieving short cycle times. This improves the efficiency of both loader and trucks and helps to keep down cost. Drives and hydraulic assemblies are, today, designed for easy exchange which reduces maintenance times. A higher degree of technical knowledge and cleanliness of the maintenance area are required than for non hydraulic equipment. This has, in the past, resulted in low availability of machines where cleanliness was not adhered to and maintenance and repairs was carried out on the quarry floor. Although these excavators are more mobile than the electric face shovels capacity rapidly falls off with increasing loading distance between loading and delivery.

Table 5 compares the features of the three types of loaders commonly found in quarry applications.

With the track type loader the dozer pushes the rock over an edge so that it can travel downwards towards the crusher site, on the lower levels, by gravity. This negates the use of trucks with the need for their associated well maintained haul roads. Narrower benches with lower stripping ratios can be planned as well as steeper gradient roads. Today dozers are used more often to remove rock from the bottom of a bench where it may impede subsequent blasting operations and in conjunction with loaders and trucks to optimize the cleaning of faces.

As the tonnages per hour loaded increases the effective cost per ton decreases. This will happen when bucket fill efficiency and swing time improves as fragmentation size decreases. Shovels, loaders and excavators were made to load broken rock not to dig it.

Table 5 Comparison of loader characteristics

Summary of Features Machine Type	**Face Shovel**	**Rating Front End Loader**	**Hydraulic Excavator**
Mobility	V Poor	Good	Poor
Break out Force	High	Low	High
Long reach	Yes	No	Yes
Versatility	V Poor	Good	Poor
Operator fatigue	Low	High	Low
Clean up ability	Nil	Good	Marginal
Load over truck end	Yes	No	Yes
Capital cost	High	Medium	High
Trade in value	Low	High	Low
Limited space operation	Good	Poor	Good
Separation of big rocks	Good	Poor	V Good
Operate in poor ground	Yes	No	Yes

7.3.2 Trucks as Haulers

In quarries the haul truck is the most economic and versatile method of moving rock or overburden from ever advancing work faces. Haul trucks range in size from 17 to over 200 tons. Those commonly found in quarries are between 35 and 80 tons. Payloads and horsepower has risen on many models in recent years and together with improved engine technology and fuel injection has reduced the unit cost of hauling. To match the system the right number of trucks is required with a bed configuration and size to make a good loading target permitting the quick and easy loading distribution. The majority of these dump trucks are of the rigit, two axle, variety.

An articulated truck, of two or three axles, is designed to work in areas requiring high maneuverability where the ground conditions are poor. They are generally of small capacity, 25 to 40 tons and there bed configuration is such that they must usually be loaded over the tail. These trucks are becoming increasingly popular in areas of selective mining where, with a backhoe as a loader, careful loading is the norm.

7.3.2.1 Matching of Loaders to Truck

Full trucks, loaded quicker, will reduce hauling costs. Tire wear is reduced as less rock is spilled on the roads. Easy loading will allow efficient scheduling of the trucks and less waiting time.

Possibly the main factor when selecting a loader is how many passes will it take to fill the hauler? An even number of passes is a must to eliminate the odd half bucket at the end or to send the hauler away without a full load. A loading cycle should be completed in two minutes. This is about the time it takes for the crusher to consume a truckload, if it is a correct match. It allows three or four swings of the bucket to fill the truck. A loader that takes two swings may result in damage to the truck's suspension as the load dropped, by the bucket, into the back will be greater than the trucks' designed capabilities. Too many swings means excessive time to fill the truck resulting in long waiting periods for the trucks to load. The optimum passes to load a truck are shown in table 6.

Too many trucks to cater for a small loader gives truck overcapacity, too few will result in delays if one truck breaks down. To cater for re-sizing of a fleet may see the need for a high lift loader. This will result in a small decrease in the bucket size and an extra pass or two.

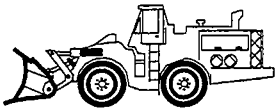
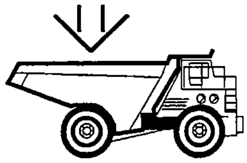
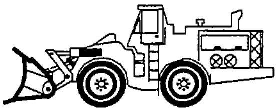
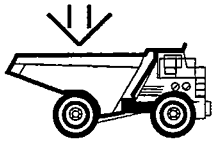
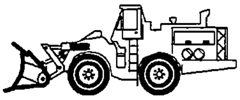
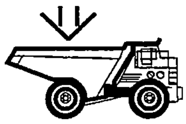
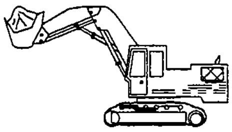
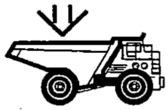
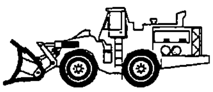
Table 6 Truck loading

Loading Machine	No. of Passes
Excavator	3 to 4
FEL	3 to 5

Figure 22 shows a match between loaders and trucks of various sizes.

These figures maximize truck usage and results in minimal loader standing time. If at the same time the loader has little maneuvering or waiting time the transport capacity is equal to the loading capacity. This is the most economical level of loading and hauling.

Figure 22: Matching Loaders to Haul Trucks

Production (t/hour)	Loader	Bucket Capacity (m^3)	Truck	Passes
1600		10.3	80 t Load 	4
1400		9.10	60 t Load 	4
800		5.40	50 t Load 	5
700		3.80	40 t Load 	6
600		4.30		5

Some manufactures publish payload ratings for their loaders. There are specific standards by which payload is calculated. If unavailable then a 'rule of thumb' is that the payload can be estimated by multiplying bucket volume by broken material density. An approximate figure for material density is 1800 kgs per cubic meter. This is the approximate weight of one cubic meter of broken granite or limestone.
For an 8 cu-m bucket: 8 * 1800 = 14.4 tons

7.3.3 Rolling Resistance

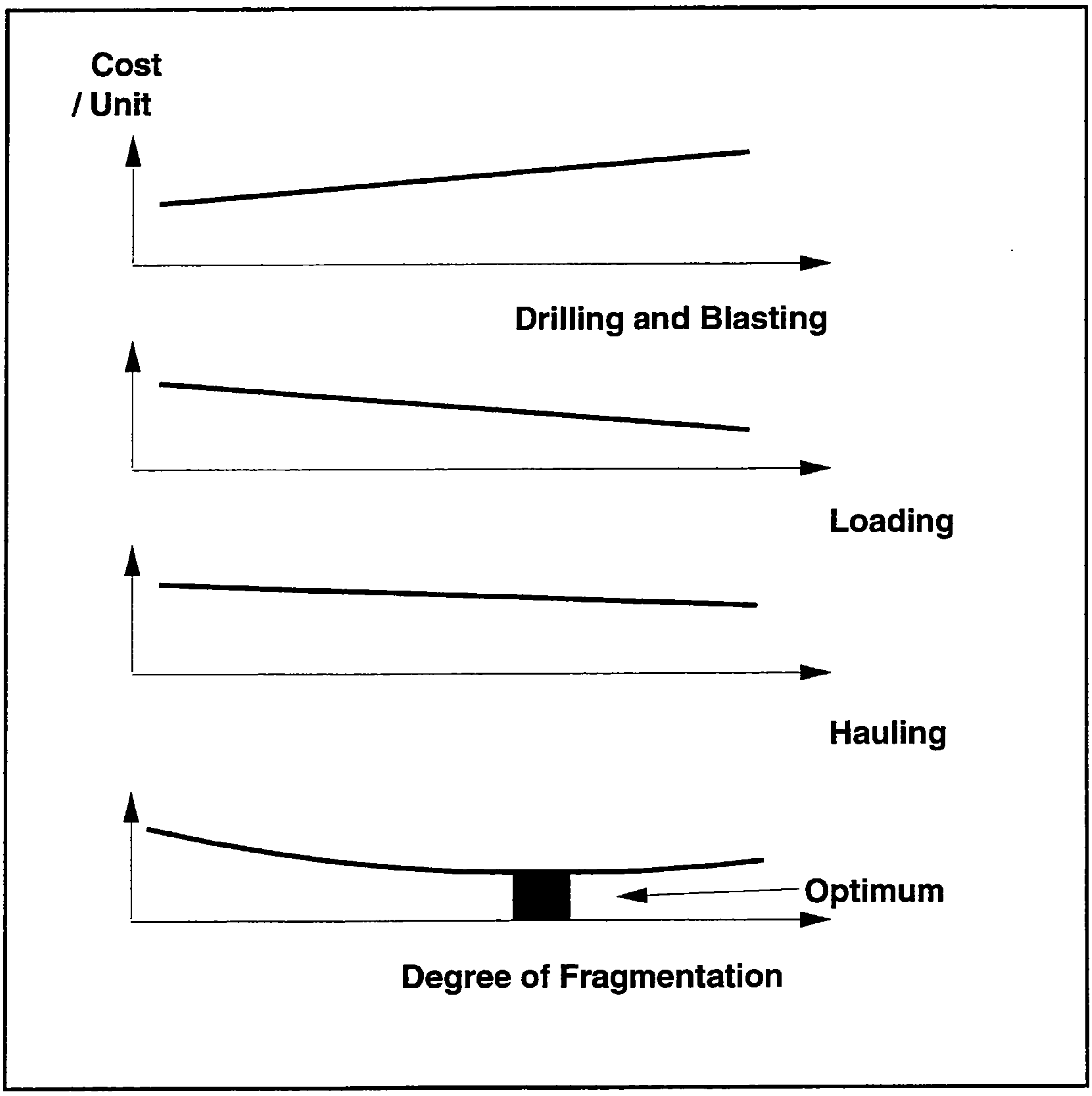
This is the retarding force of the ground against the wheels of a vehicle. This force must be overcome before a vehicle can move. Many things determine the rolling resistance or the retarding force acting against a rolling wheel. Among the most important are:

1) Internal friction
2) Tire flexing
3) Penetration of the wheel into the surface
4) Weight on wheels

These effects should amount to about 2 % of gross vehicle weight. This is the rolling resistance factor of a truck traveling on a concrete highway. Shown in figure 23 is the rolling resistance for a 60 ton truck traveling over various roadway surfaces. it can be seen that the resistance can increase dramatically if the surface is not well maintained.

Rolling resistance does not however affect tracked type vehicles. This is because they carry their roadway with them and steel is always hard and smooth. Only internal friction has any affect on these vehicles.

Figure 23: Rolling Resistance



7.3.4 Grade Resistance

It is the force of gravity which must be overcome when going uphill. It acts against the total weight of any vehicle. Grades are usually expressed as percent (%) slope, which is the ratio between vertical rise or fall and the horizontal distance over which this occurs. When the grade is uphill, adverse, the effect is to demand more power. If downhill, favorable, then it is a helping force producing additional pull to propel the vehicle. This downhill effect is commonly called grade assistance.

7.3.5 Effective Grade

This is the total resistance expressed as the grade which would have to be negotiated to get the equivalent resistance to motion.

Since Total Resistance = Rolling Resistance + Grade Resistance

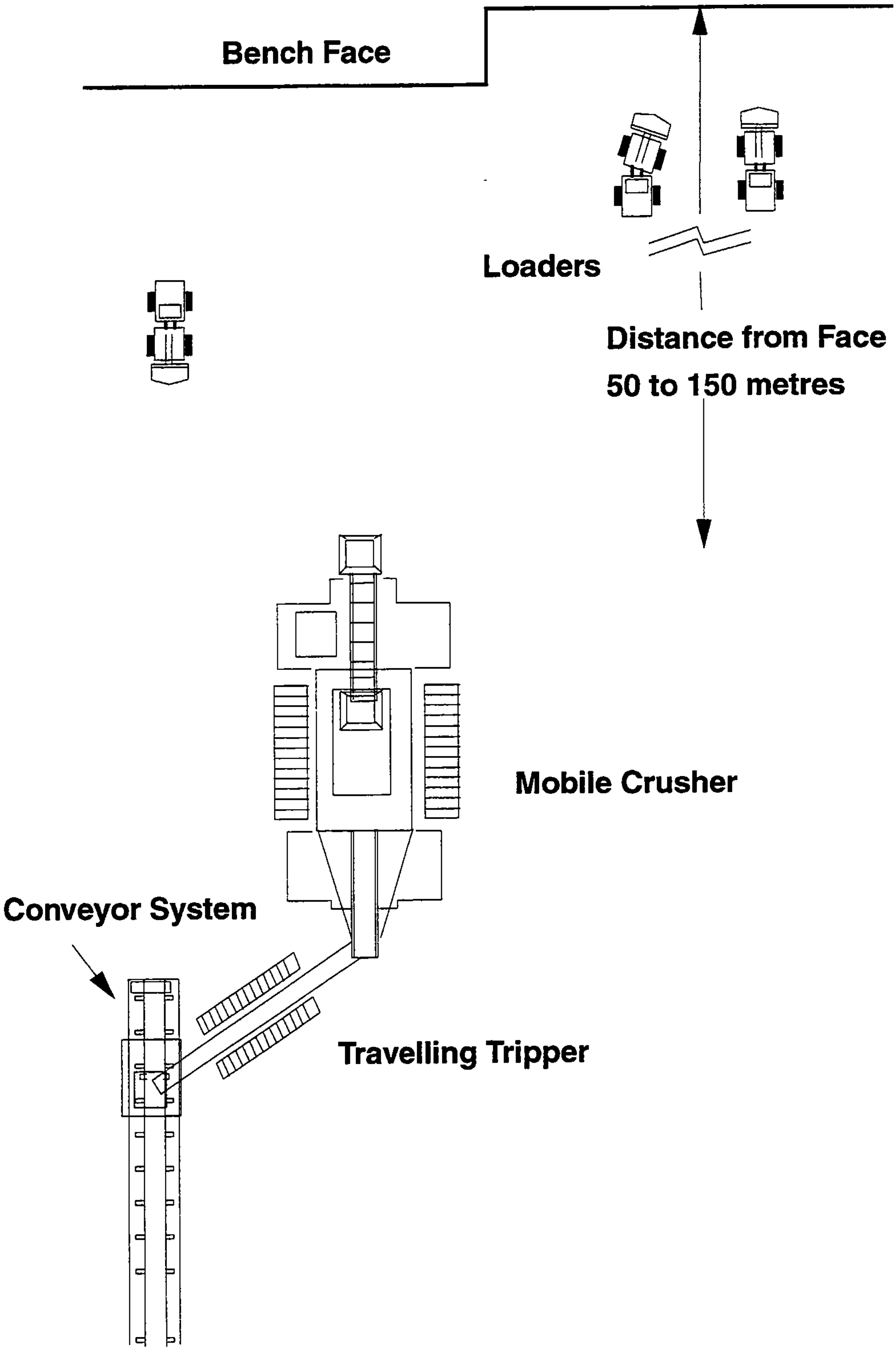
and
1 % of grade produces a grade resistance of about = 10 kgs/t of vehicle weight

Effective Grade % = Rolling Resistance (kgs/t) / 10 + % Grade

7.3.6 Cost Relationship

From figure 24 it can be seen that as the cost of drilling and blasting increases so the cost of loading and hauling decreases. This is mainly due to the easier loading conditions that exist when the fragmentation of the muckpile is finer. Loading is easier, there is a smaller percentage of oversize rocks in the muckpile, and hauling is quicker as the trucks do not have to wait at the crusher while blockages are cleared, to give two examples. Studies have proved that over blasting of the rock, resulting in fines fragmentation, will actually decrease the overall cost of the quarry operation.

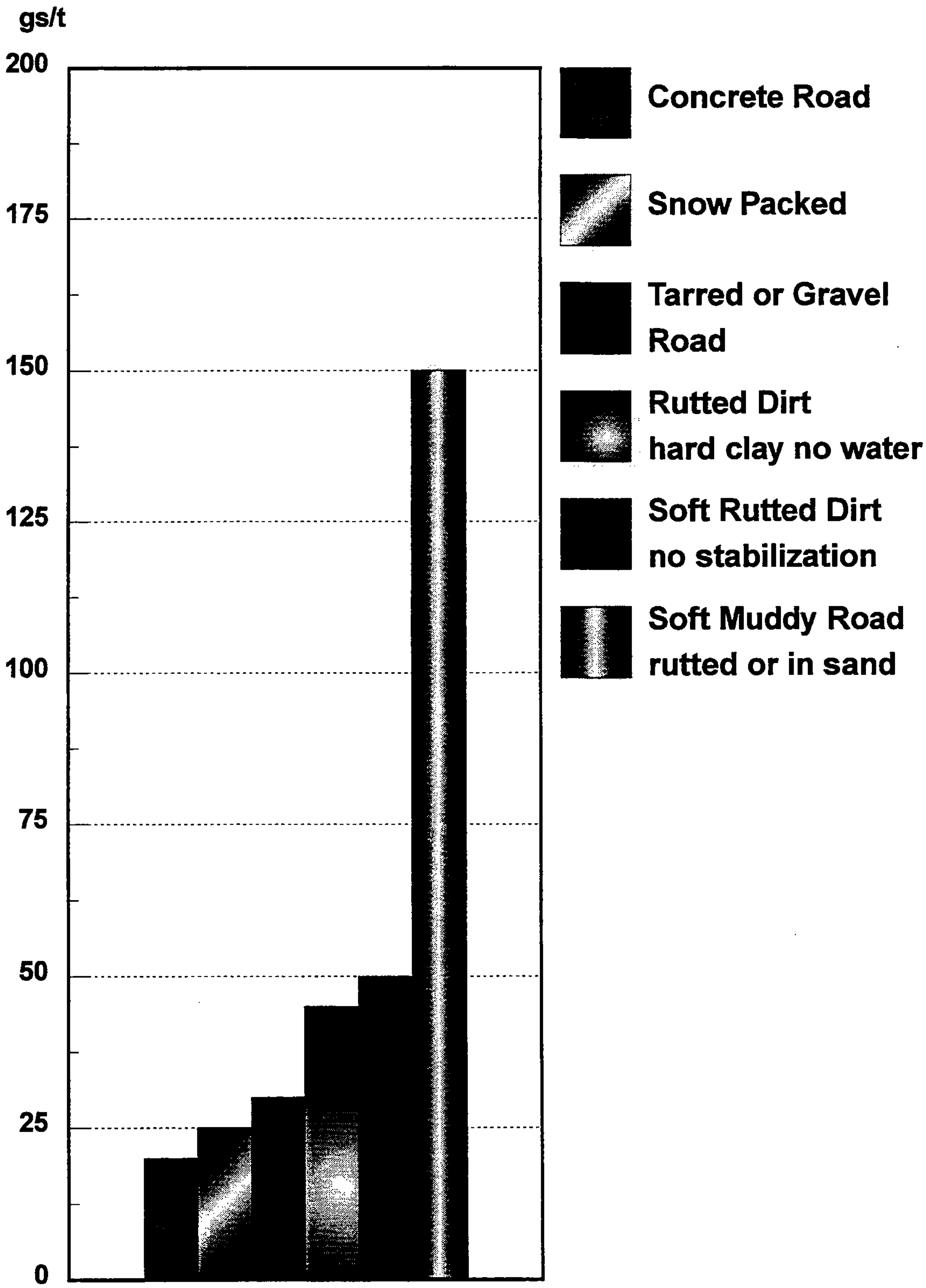
Figure 24: Cost as a Function of Degree of Fragmentation



7.4 In Pit Crushing

Transport between the loader and the fixed crushing plant is nearly always by a discontinuous transportation method, trucks. This can be considered a disadvantage in that either end of the transport system is in continuous operation. As the quarry faces are always advancing the site of the crusher does not remain optimum for long. To facilitate this an ever increasing number of trucks is required to maintain production with associated increased costs. An answer to this problem has been to install in-pit mobile crushers which considerably reduce the need for trucks. Rock is loaded from the muckpile to the primary crusher and is then fed to a conveyer belt to be transported to the plant. The crusher in the quarry is usually mobile so that it can follow the advancing faces. This is made possible by the addition of conveyor belts when the unit is moved forward. Figure 25 sketches a typical system. Secondary and tertiary crushing systems are, if required, on a fixed site outside the quarry. The capital investment to purchase a mobile crusher is higher than for an average fleet of trucks. However the savings in manpower and increased output from the loading equipment reduce the overall working costs and make the unit justifiable.

Figure 25: Mobile Crushing System



8. CONTRACTING OUT THE QUARRY OPERATION

There is a trend to contract out the quarrying operation in our industry. It has resulted from the belief that experience in quarrying is limited and that capital investment can be saved from not having to purchase mobile equipment. This initially manifest itself in the contracting of the drill and blast operation where it was found to be cost effective. Out-sourcing the entire quarry operation appears to have several advantages.

- payment is for guaranteed work done
- it is possible to inspect the work before payment is made
- competition between contractors can lower the cost
- provides additional expertise in the day-to-day running of the operation
- eliminates peak vehicle requirements

It is necessary to supervise the contractor as in all operations and this can be done in a number of ways. The most common method of recording tonnage delivered to the crusher is to send every truck load over a weighbridge. However this is both time consuming and wasteful of the contractors resources. It is more common to pay a contractor for the amount of clinker produced. This is a much more reliable indicator of the tonnage moved. The requirements of the plant should be detailed and the following points should form part of the contract:

- payment on the quantity of clinker produced
- limits on the size of material delivered to the crusher; preferably a size distribution of the incoming raw material
- limits on the quality acceptable
- tonnage on a daily basis
- aspects of the companies safety policy
- an agreed environmental protection policy

Included in the working cost of any out-sourcing must be that cost for the owner to supervise and manage the operation. In many cases the regulations specify the responsibilities of the owner and specific mention is made of his duties which cannot be given to the contractor in a quarrying operation. Due in part to this fact the long term cost of operation is usually cheaper if not contracted out.

9. RECLAMATION

Public acceptance of quarries is, in part, dependent on the creation of a suitable after use. A range of uses is possible from lakes to development of urban housing schemes. All to often reclamation of an old quarry is unrealistic or unfeasible.

9.1 Restoration to Agricultural Land

This is probably the most common restoration option. The techniques used are based on a complete understanding of the soils, geology and hydrogeology of the site. The various soil horizons will be stripped separately and stored for future use. Handling has to take into account damage to the soil that occurs when moving it. When replaced a five year care program of cropping ensures that the restored farmland is productive. The quarry is usually filled with overburden or imported wastes to bring it to its former outline. The use of domestic and industrial wastes is not recommended as seepage of ground water could lead to contamination of rivers and lakes, and also have a detrimental effect on the reclamation project. High costs of returning worked out areas to farmland, occasionally several times the market value, has led to it being developed for other uses.

9.2 Forestry

Restoration to woodlands is usually only possible on the worked out floor of quarries or on sites filled with inert material. There are also problems associated with drainage due to compaction by earth moving equipment. Artificial ridges and furrows can overcome this. Tree planting restores the balance of tree loss caused by the demands of agriculture.

9.3 Recreational

Many wet pits have been turned into small lakes for sailing and boating. If a constant supply of oxygen rich water is available a quarry site could be turned into a fish farm. Where the surface is dry, playing fields or golf courses have been established. The same principles of reclamation apply as for agricultural use. If this type of reclamation is adopted then it can be highly profitable to the owners of these worked out quarries.

9.4 Nature Conservation

These sites tend to happen rather than be designed. Many old quarries become filled with rain water and attract wildfowl, especially if in a remote, sparsely populated area. Once established they can become important nature conservation areas.

9.5 Urban Uses

Many quarries came into existence because of their proximity to urban areas. With the rapid growth in population in the last few decades they now find themselves surrounded by urban areas where once was open country. Few problems are posed by the use of quarry floors for buildings. The rock base is stable for foundations, only the slope angle will have to be modified. If domestic housing is proposed then a supply of topsoil will be needed to establish gardens. Unsightly factories can be hidden from general view by placing them in old quarries.

Building on fill is not to be recommended. Subsidence may affect buildings and there could be problems with gases emanating from the fill. In some cases methane explosions, with gases from fill, have occurred.

With a positive after use a quarry is much more likely to be tolerated by a community. Reclamation should be ongoing as the quarry progresses. Even a line of trees, to shield the working from view, will assist in its acceptability.

10. THE FUTURE

With rising costs affecting profits the need to increase productivity to minimize this means that the move towards mechanization and the use of computers will increase. However the cost of replacing any machinery incapable of mechanization or new computer hardware must be taken into account. The benefits can include, reduced operating costs, maximum equipment utilization and increased safety for workers.

Some of the areas in the quarry where increased productivity can be achieved are:

10.1 Drilling

Drilling rigs can be computerized leading to more accurate spotting of holes and more accurate drilling. Drillholes will be accurately laid out with use of computer programs, and a disc given to a rig operator who inserts it into an on board computer. This will line up the rig and collar the hole independently of the operator, removing him from the error equation in the optimization of the drilling and explosive costs.

10.2 Explosives

The use of mobile manufacturing plants in the manufacture and delivery of explosives. Instead of having to build and service magazines for the storage of explosives and accessories they will be delivered to the blast site, from the manufacturer, at a time requested by the blaster. The composition of the explosive can be varied to suit ground conditions on site, even within a single hole.

10.3 Accessories

Electronic detonators which contain a microchip to give extremely accurate timing. The scatter in the actual detonation time of present delay detonators can cause either an overlap of shots of different delays detonating at the same time. This gives rise to increased noise and ground vibration levels as well as poor blasting results.

10.4 Controls

Computer aided survey packages to profile bench faces and to plot drillholes accurately. Actual burdens and spacings anywhere in the length of the holes could be plotted and more accurate estimation of the explosive required made. Monitoring, closely, the actual lengths off holes can prevent overdrill with resulting poor quarry floors.

10.5 Planning

Traditionally planning has been divided into short term, one to three months, and long term, two to five year, planning. Every year a week is spent updating these plans and explaining the difference between actual and achieved. The use of computer aided planning will keep a closer control on the raw materials production and optimize the extraction in both, quantity and quality. Planning can be updated whenever necessary as daily results can be quickly entered.

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HAUL ROADS

- A truck is productive if it is hauling rock.
- Highest productivity only on best haul roads (cycle times).
- The main roadway should be established in ground that requires a minimum of maintenance.

HAUL ROADS

- Features:

Safety and Health

➢Long life
➢Minimum bends
➢Wide as possible
➢Good surface
➢No potholes
➢Crossfall

$$$$$$

Productivity

HAUL ROADS

- Enemies of the roads:

➢Water
➢Too heavy payload
➢Inclination
➢Frequency of trucks (Swinging trucks)

HAUL ROADS

- Enemies of the roads:

HAUL ROADS

HAUL ROADS

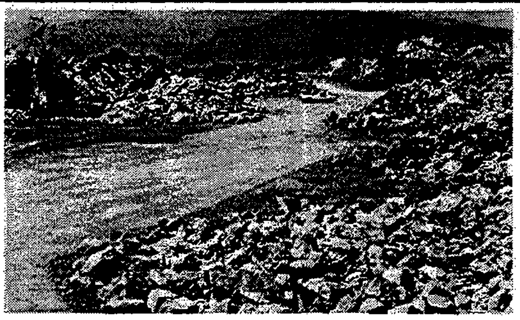
- Affects costs:

➢Tires (5 to 15k US$ each)
➢Fuel (rolling resistance from 11 to 3% => 1/2 fuel)
➢Maintenance on equipment
➢Productivity due to velocity of equipment
  - ➢ Down time / Death time
  - ➢ Overtime
  - ➢ More equipment necessary
  - ➢ More labour necessary

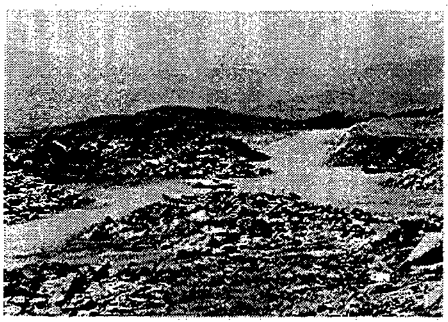
HAUL ROADS

➢Last but not least: Safety of Personnel

HAUL ROADS



HAUL ROADS



HAUL ROADS

HAUL ROADS

HAUL ROADS

- Geometry of a haul road:
  - ➢Number of lanes
  - ➢Safe distance
    - ➢ Drivers reaction time = 2 sec.
    - ➢ Allowance for error = 5 sec.
  - ➢Road width
  - ➢Super-Elevation
  - ➢Safe sight distances
  - ➢Camber
  - ➢Bend

HAUL ROADS

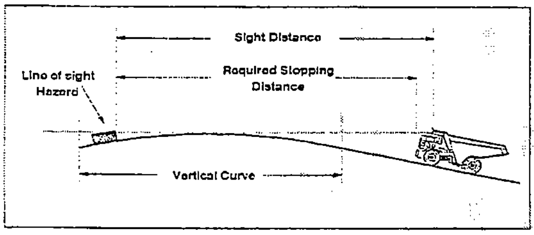


Figure 3 Vertical curve Line of Sight

HAUL ROADS

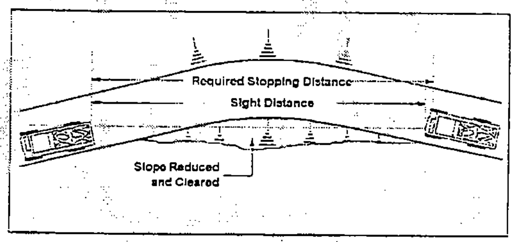


Figure 4 Horizontal curve Line of Sight

HAUL ROADS

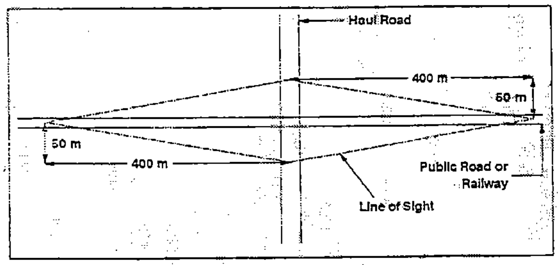


Figure 5 Line of Sight at a Level Crossing

HAUL ROADS

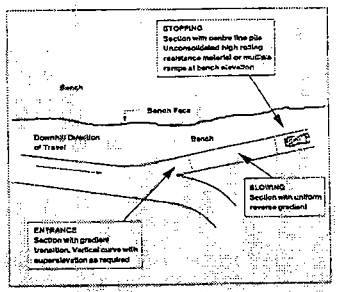


Figure 6 Runaway Precautions

HAUL ROADS

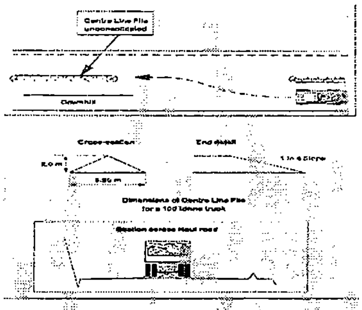
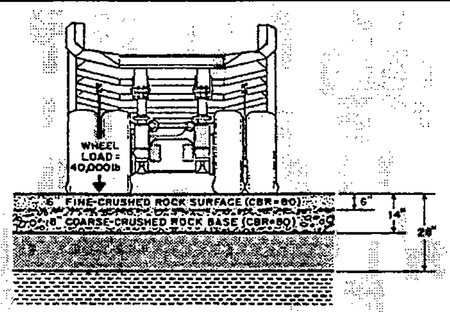
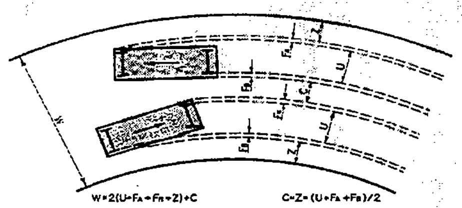


Figure 7 Centre Pile Retarder Layout

HAUL ROADS



HAUL ROADS



$W = 2(U + F_A + F_R + Z) + C$ $C = Z = (U + F_A + F_R)/2$

U = Track width of vehicle (center-to-center tires), feet
F_A = Width of front overhang, feet
F_R = Width of rear overhang, feet
C = Total lateral clearance
Z = Extra width allowance due to difficulty of driving on curves, feet

HAUL ROADS

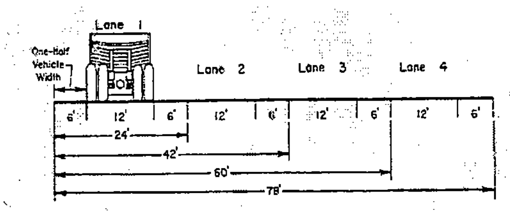
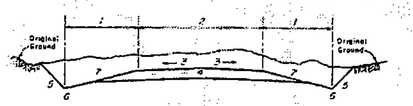
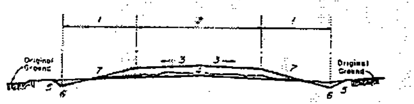


FIGURE 21. - Typical haulageway sections for 12-foot-wide vehicle.

HAUL ROADS

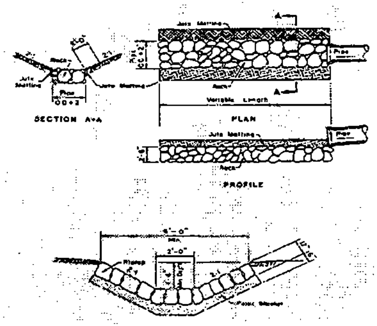


TYPICAL CUT SECTION



TYPICAL FILL SECTION

HAUL ROADS



HAUL ROADS

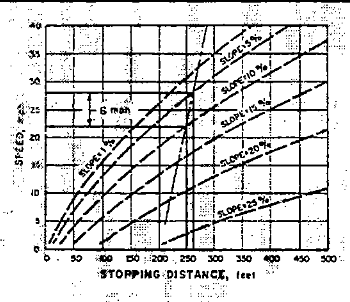
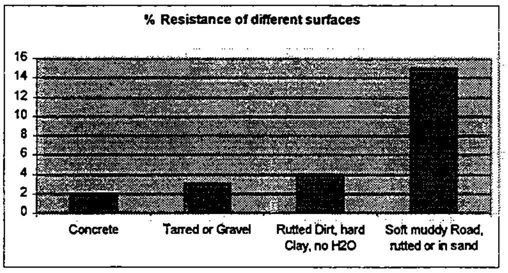


FIGURE 7. - Effect of grade reduction on stopping distances.

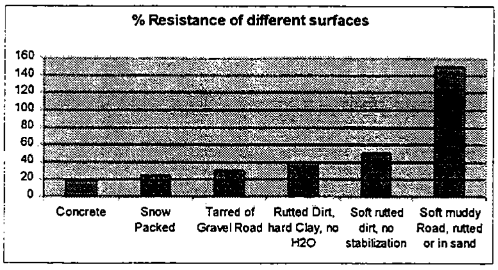
HAUL ROADS

- **Rolling resistance**



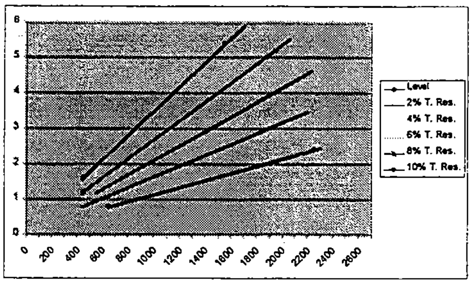
HAUL ROADS

- Rolling resistance



HAUL ROADS

- Rolling resistance and truck travelling times

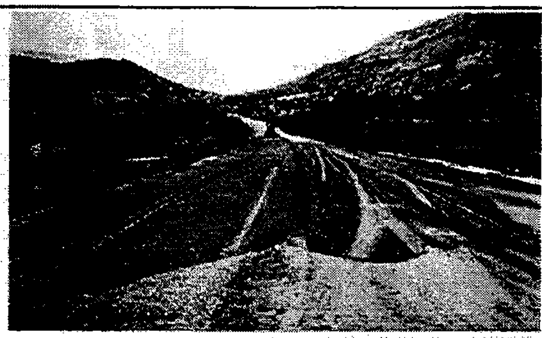


HAUL ROADS

- Costs of "Rolling Resistance"
  - ➤An increase from 3 to 11% means:
    - ✓ 50 % Fuel increase
    - ✓ decreased productivity per truck
      - ➪ therefore more truck time is necessary

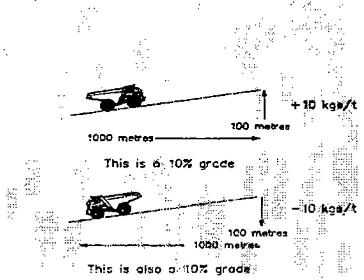
Σ Double Fuel Bill !!!!!!!!!!!!!!!

HAUL ROADS

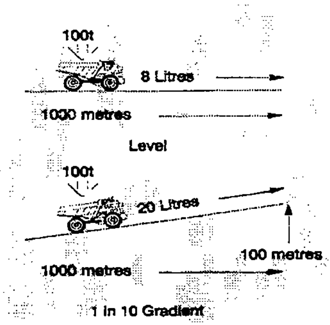


HAUL ROADS

1% Grade produces a hindering or helping force of 10 kgs/t of vehicle weight



HAUL ROADS



HAUL ROADS

- Competition to the Haul Roads:
  - Conveyor Belts
    - Quiet
    - Not affected to the weather
    - Dust contained
    - Labour efficient
    - Spillage minimal
    - Energy efficient
    - Continuous transport system

HAUL ROADS

- Competition to the Haul Roads:
  - Conveyor Belts

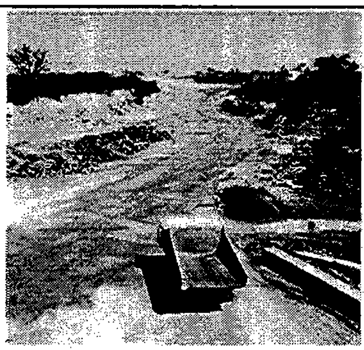
Maximum size << compared to truck loads
Mobile crusher has to be installed

No haul road is necessary

HAUL ROADS

HAUL ROADS

HAUL ROADS



HAUL ROADS